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FACULTY OF MINING AND GEOLOGY
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DIPLOMA THESIS

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Bc. Leon Matheus

VYSOKÁ ŠKOLA BÁŇSKÁ – TECHNICAL UNIVERSITY OF OSTRAVA
FACULTY OF MINING AND GEOLOGY

Proposal for Modernization of Blasting Works in Austin Detonator Powder Services Company

DIPLOMA THESIS

Author: Bc. Leon Matheus
Supervisor of the diploma thesis: doc. Ing. Milan Mikoláš, Ph.D.

Ostrava 2016

DEDICATION

I dedicate my diploma thesis work to my late mother, Magtalena Etete Kheiamses, may her soul rest in internal peace.

Affidavit:

I hereby declare that this diploma thesis work is my own unaided individual work. Proper attribution has been given to all outside sources. It is being submitted for Ing. Mining Engineering at VSB – Technical University of Ostrava, Czech Republic.

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ABSTRACT

The aim of this diploma thesis assignment was to design a new blast design for Mokra Quarry. Since Mokra Quarry was not a hypothetical mine but a mine with existing operations that incorporates current blasting works the new design was to serve as means to modernize the current blasting works. Blast design is important because good design produces desired fragmentation and reduces cost related to material handling or secondary blasting. The modernization of blasting works at the quarry required detail assessment of current blasting practices that are undertaken. Therefor a visit to Mokra Quarry was contacted to obtain information of current blasting parameters. During the field visit information on geology, bench geometry, explosives, detonators, blast pattern and firing system were recorded. Mine workers on site assisted with logistics and technical information. Based on the information gathered planning model was formulated that incorporates input data and predicted output (results). The thesis has three main sections of importance (1). Evaluation of current blast design and parameters (2).Proposed new blast design and (3). Comparison and evaluation of current and proposed blast design.

The diploma thesis finding were that the current blast design was not sufficient in comparison with the proposed blast design because of poor fragmentation **(i.e. excess fines)** and high cost **(i.e. secondary blasting**).Therefor it was recommended to amend or re-evaluate the current blasting works to benefit the quarry.

KEYWORDS: Blast Design, Fragmentation, Blast Pattern, Modernization, Bench Geometry, Geology, Explosives, Detonators and Firing System

CONTENTS

INTRODUCTION

Breaking rock is the first step in mining process and is done so that the material can be moved to different locations for waste dumping, stock piling or for further processing. Explosives are used in situation when the rock cannot be free-dug or continuously mined due to its geology setting. [24]

Surface mining uses explosives for primary blasting that breaks the rock out from solid and secondary blasting is used to reduce large blocks to smaller size for easy handling or due to client requirement. Secondary blasting is considered to be expensive, time consuming and slow the productivity of the mining operation. To reduce the cost from secondary blasting most mines design a primary blast that is good to eliminate or minimize secondary blasting. [14]

The main purpose of blast design is to ensure good blast with good fragmentation results to reduce material handling cost. Blast design can either be new or optimization of old design. Some factors to be taken into account when designing a blast design include bench geometry, geology, loading/hauling arrangements, explosives and borehole diameter.

The purpose of this thesis is to modernize blasting works at Mokra Quarry, with primary objective of designing a new blast design that can be feasible. In order to achieve the objective comprehensive methodology was developed **(i.e. descriptive and analytical methods)** to gain valuable information that help in the design of the new blast. Planning model was formulated to serve as a road map throughout the project execution. In the thesis the current blast design and parameters are analysed and new design developed and comparisons made to rectify defects if need possible. The following ***chapters*** form the basis of the investigation;

a) *Chapter **1**: Mine Overview*
b) *Chapter **2**: Literature Studies*
c) *Chapter **3**: Project Investigation*
d) *Chapter **4**: Current Blast Design*
e) *Chapter **5**: Proposed Blast Design*
f) *Chapter **6**: Comparison and Evaluation of Blast Designs*
g) *Chapter 7: Conclusion and Recommendation*

1 MINE OVERVIEW

1.1 Mine Background

Historically Mokra Quarry (as indicated in Fig.1) mining operations have been ongoing since 1968. The quarry is located in village Mokra – Horakov district Brno-venkov, Czech Republic at an altitude of 410m. Mokra Quarry is the third biggest quarry in Czech Republic with surface mining area of 2 659 881m^2 and active mining area of 1 100 000m^2. Primary purpose of the quarry is to supply raw material to limestone and cement plant. [27]

Figure 1: Mokra Quarry (author's field photo)

1.2 Geology

Mokra quarry has two main prominent rock types that are limestone and clay, although significant amounts of sand, loam, gravel and schalstone are found in a typical rock matrix. Western part of the quarry is dominated by Devonian Limestone with high purity, up to 96 – 97% $CaCO_3$ contents. The main ingredient in the Devonian Limestone is Vilemovice Limestone. The Vilemovice Limestone is very fine – grained, thickly tabular to massive of light grey colour. [10]

The thickness of the limestone varies from moderate to high thickness and primary used for lime production. Middle part of the quarry dominated by low purity limestone which contain silica and aluminium that are used in cement production. Figure 2, below indicates typical geological profile in the western part of the quarry and is similar in other mine areas.

Figure 2: Schematic of Geological Profile (author's illustration)

1.3 Production Logistics

1.3.1 Production

The primary means of production at the quarry is achieved using drilling and blasting. Amount of material (ore) required depend on the building industry but the quarry produced approximately 5000t/day with target ore production being 1.5million t/year with 12% losses included. [27]

The quarry has unique set-up that could be define as less waste cost operation because low purity limestone is not considered waste but is used in cement production and high purity limestone being used for the production of lime.

1.3.2 Ore - Handling

A large range of loading and hauling equipment types are available and their selection can depend on material (ore) characteristics, transport route (grade, length and curves), manoeuvrability, mine road surface and mine production requirement. Mokra Quarry, understanding their mining environment has selected their ore handling equipment based on the factors mentioned above. A wheel loader (FEL) is used to load the blasted material into 60t dump trucks (Komatsu or Caterpillar). Wheel loader are ideal for loading loose fragmented rock. After loading the material (ore) is transported to the crushers for further fragmentation to the desired size. Figure 3 below indicates wheel loader loading a dump truck at a quarry. It must be noted that figure 3, is not picture from Mokra Quarry but is used as illustration.

Figure 3: Wheel Loader and Truck [6]

1.4 Fragmentation

1.4.1 Material Size

The material size in the quarry before fragmentation depend on environmental conditions thus as a rule of thumb during snow or rainy conditions boulders are preferred and during better or sunny days smaller blocks are desired. Generally limestone is preferred in boulders (40 – 150mm) for lime production and for cement production the size of fragmented material does not play significant role.

1.4.2 Comminution

Comminute by oxford dictionary definition means ***"reduce to small fragments"*.** It is particle size reduction of materials. Comminution may be carried out on either dry materials or slurries. Crushing and grinding are the two primary comminution processes. Crushing is normally carried out on **"run-of-mine"** ore, while grinding (normally carried out after crushing) may be conducted on dry or slurried material. In comminution, the size reduction of particles is done by three types of forces, compression, impact and attrition. Crushing is a dry process whereas grinding is generally performed wet and hence is more energy intensive. [20]

Mokra Quarry for the purpose of comminution uses a gyratory and a hummer crusher due to different production requirements. Gyratory (as indicated in Fig.4) crusher is used to crush boulders of size 40 – 150mm for lime production. [27]

The gyratory crusher is suitable because it is primary crusher and can handle large capacity. This type of crusher is also suitable for slabby feeds and slow compression of crushing head helps to limit fines generation. [20]

Figure 4: Mokra Quarry Gyratory Crusher (author's field photo)

Hammer crusher (as indicated in Fig.5) is used in cement production by crushing material size < 40mm. This type of crusher is ideal because it can crush material with middle or less than middle hardness into secondary or fine granularity. Main advantage of this type of crusher is high production and high reduction ratio, low power consumption, homogenous particle size, simple compact and light mechanical structure and minor cost per the processed material unit. [20]

Figure 5: Mokra Quarry Hammer Crusher (author's field photo)

1.5 Reclamation

Mining inevitably disturbs land therefor modern mines reclaim the surface during and after mining is completed. Reclamation is a process of returning land to its original or better condition after mining is completed. It should be part of mining activity and it can be divided in four phases: [13]

- Preparation and design phase.
- Mining- technical phase of reclamation.
- Bio-technical phase of reclamation.
- Post reclamation phase.

The objective of reclamation is to return the land and watercourses to an acceptable standard of productive use, ensuring that any landforms and structures are stable, and any watercourses are of acceptable water quality. Reclamation typically involves a number of activities such as removing any hazardous materials, reshaping the land, restoring topsoil, and planting native grasses, trees, or ground cover. [32]

Mining has long tradition in the Czech Republic and still is an important part of the country's economy although recently its economic importance has been decreasing. However it still has significant impact on the landscape and nature. Mokra Quarry is situated in a valuable area of Southern part of Moravian Karst and from the biodiversity point of view unique because of its limestone bedrock, which is connected with the rich communities of deciduous forest and dry grasslands. [18]

A large amount of the quarry floor also consists of aquatic and wetland vegetation, which is habitat for many species of animals. [2]

Mokra Quarry under Czech Republic mining legislation, Act No. 44/1988 Sb. (Mining Act) and Act No. 61/1988 Sb. with Decree No. 104/1988 Sb. and No. 52/1997 Sb. [13], has made reclamation part of mining activity and ensure biological reclamation as standard practice to reclaim mined out areas and pillars left behind thus maintain the rich biodiversity of the mined out areas over time. Figure 6 below indicates reclaimed mining area with vegetation clearly visible covering previously mine bench.

Figure 6: Reclaimed Mining Area in Mokra Quarry [27]

2 LITERATURE STUDIES

The literature studies in this thesis is intended to broaden the understanding on blasting works. In the first section evolution of quarry mining is discussed, second section discusses the importance of blast design, design parameters, detonators and explosives. In third section a case study from limestone quarry in the Philippines is analysed comparing effect of two different firing systems. The three sections emphasized in the literature studies are valid because the modernization of blasting works at Mokra Quarry will require understanding of concept used in blast design and work done at other quarries.

2.1 Quarry Mining

Methods of extracting stone and other materials from quarries have changed (as indicated in Fig.7*)* since the first quarries were mined in the Aswan area of Egypt. The earliest quarries were mined with hammers, pics and chisel made of stone or metals such as bronze and iron. Even communities that did not have stone buildings created quarries. The Lakota culture of the Midwest region of the United States and Canada did not quarry stone to build monuments or houses. At a site in Pipestone national monument, in the state of Minnesota in United States, they quarried for stones to make calumets, or ceremonial smoking pipes. Quarrying material for use in building materials was much more work. Stones had to be carried or dragged out of quarries manually. Stones could also be hauled with pulley systems involving ropes and moveable wooden tracks or sleds. This process often involved thousands of workers, slaves or community. [29]

However in the in modern day a quarry is a type of open pit mine used to mine building materials such as dimension stone, rock, construction aggregate, riprap, sand and gravel. Today quarry mining operations used drilling equipment (i.e. hummer drills), blasting equipment and hauling equipment (FEL & ADT's). Industrial drills with diamond tips are used to cut into hard rock. Some quarries used explosives as means for primary fragmentation. Advantages of quarry mining are low capital cost and low mechanization, easily accessible and well suited for small deposits, stable wall and benches, generally no bank support required as there little chance of slope failure and high selectivity and good safety operations. [30]

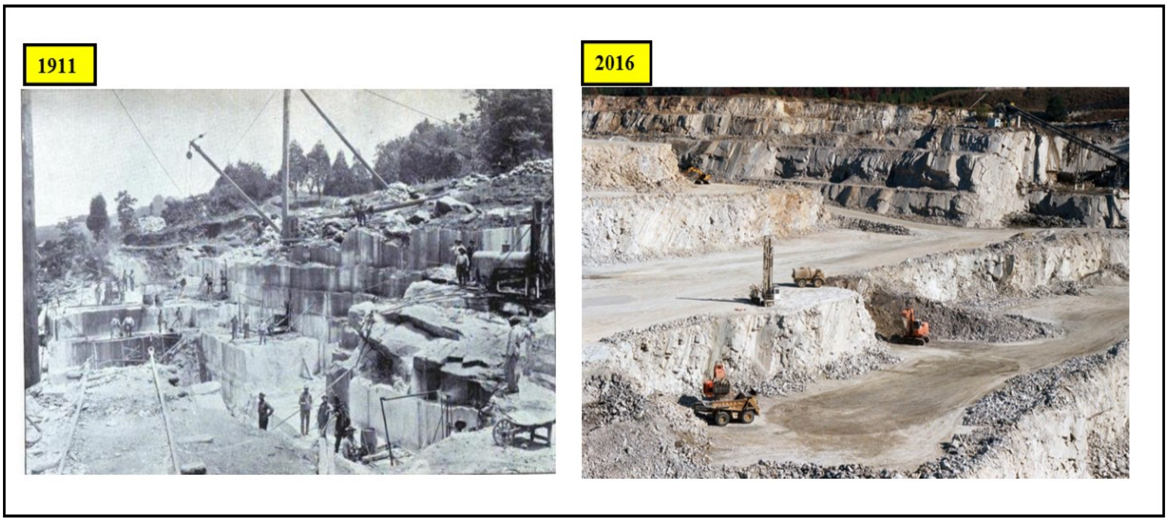


Figure 7: Changes in Quarry Mining [29]

2.2 Blast Design Concepts

2.2.1 Blast Design Necessity

When mining competent rock the production cycle starts with drill and blasting. Improper blasting practices can result in economic and operational problem for the mine particularly if the operation is close to residential area or near some form establishment. The aspects of safety and environmental factors should also be considered. To eliminate economic, safety and environmental factors a blast must be designed to achieve desired objectives with minimum or no negative effects. A well designed blast is a controlled blast with organized firing systems and drill patterns but with consideration of desired fragmentations, ground vibrations and fly rock issues. Primary purpose of blast design is to distribute explosive energy in such way that desired fragmentation and muck pile displacement is achieved. [1]

When blast holes are fired independently, a cylindrical 'plug' of broken ground is created around each hole before movement of the burden take place. The diameter of the 'plug' is determined by the pressure of explosive gases and the time for which they act in the radial cracks growing from the blasthole. Release of pressure occurs by venting through both the stemming and via radial cracks and fissures to the free face. If burden is small, gas release very quickly and its unused energy is spent in heaving the broken burden and when burden is large it promotes formation of longer cracks and greater diameter plug of fractured ground with minimum heave. [14]

It is impossible to establish a blast design numerically thus certain empirical rules must be used to enable the blasts to be numerically analysed. Controllable variables (burden, spacing, borehole diameter, bench height, stemming etc.) as indicated in figure 8. and uncontrollable variables (water, structural discontinuities, material strength etc.) form part of the empirical rules that are established in the design of the blast and outcome such as fragmentation size and muck pile profiles can be approximated and thus ore – handling system can be established.

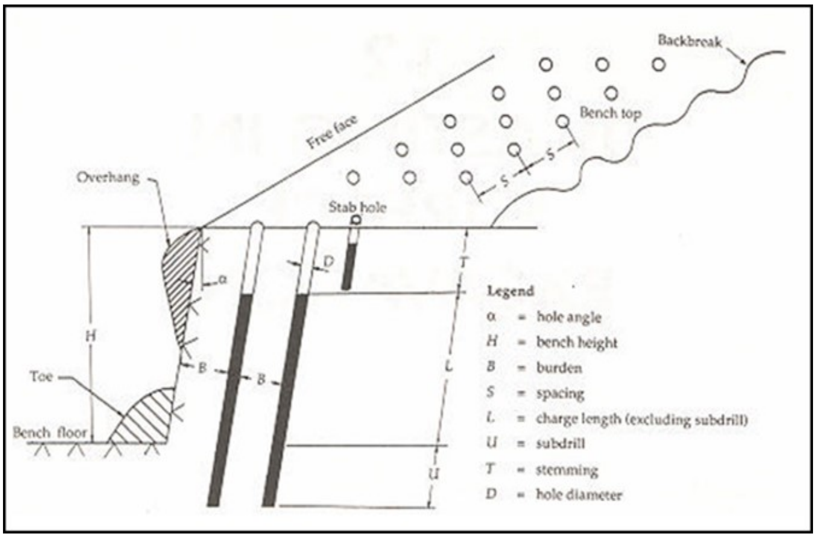
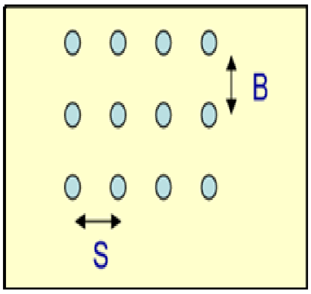
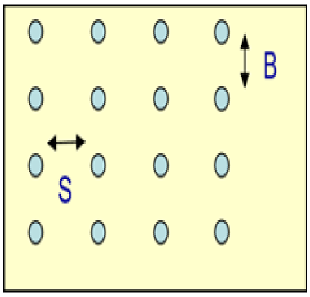
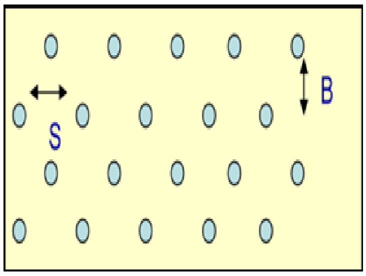


Figure 8: Blast Pattern Parameters [31]

2.2.2 Blast Patterns

Table 1 below indicates commonly used blast patterns in Quarry mining that is Square, Rectangular and Staggered. Generally a staggered pattern is used for row or diagonal firing, where the holes in one row are before the holes in the row immediately behind them. The square and rectangular blast patterns are used for “V” (Chevron) or echelon rounds. The two common firing systems will be discussed in the case study that forms part of the literature studies.

Table 1: Blast Patterns [12]

Square Pattern	Rectangular Pattern	Staggered Pattern
Drilled spacing that are equal to drilled burdens 	Drilled spacing that are larger than drilled burdens 	1. Drilled spacing of each row are offset such that the holes in one row are positioned in the middle of the spacings of the holes in preceding row. 2. Drilled spacing are larger than the drilled burdens. 

2.2.3 Detonators

The literature discussed below focuses on the general characteristics of detonators and non-electric detonators in particular since the blast studied in the thesis uses non-electric detonators provided by Austine Powder Company. Any explosive requires stimuli like shock, friction or flaming for it to blast. The devices used to facilitate these operations are known as initiating devices or detonators. In general context a detonator (as indicated in Fig.9) is a capsule of sensitive explosive (Cu, Bronze or Al), with outer diameter of 5.5 – 7.5mm, accompanied by varying length depending if it is instantaneous or delay type. The strength of a detonator is measured on the quantity of base charge and A.S.A charge quantity; the detonators are designed as detonator no.1 to no. 8 or more in the order of increasing quantities of two charges mentioned in above text. Thus in practical No. 8 cap produces much stronger pressure pulse than no. 6 cap. No. 6 detonator contains 0.35gms of A.S.A mixture and 0.25gms of PETN or tetryl. No. 8 carries large charge, 25% more than No. 6 and is used in the blasting of hard rocks. The method of initiating the charge can be safety fuse, as in case of plain detonator or by fuse head as in case of electric detonator. [3]

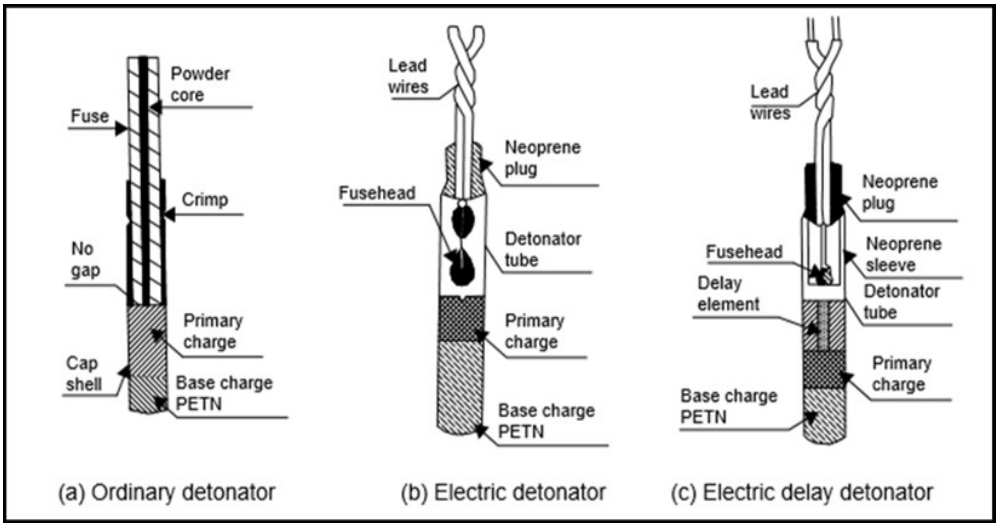


Figure 9: Types of Detonators [3]

Non –electric initiation systems have been used in the mining industry for many years. Cap and fuse was the first method of non-electric initiation. The systems used in the current mining industry consist of detonating cord, shock-tube detonators or a combination of the two. The advantages of non-electric initiation systems are, it does not get affected by stray electric current and radio frequencies. However the setback are the susceptibility to accidental initiation by lightning strike and accidental detonation by heat or impact because they contain sensitive ignition charges and base charges. [9]

In addition the system provides better fragmentation thus allow mines to decrease time of truck loading by up to 10-15% thus reduce cost. Other advantages include reduction in ground vibrations and better muck pile profile after blasting is done.

2.2.4 Explosives

The literature discussed below focuses on Gelatin and ANFO explosives since the blast studied in the thesis uses Dynamite Perunit Explosive and ANFO Austinite 3 eco provided by Austine Powder Company. Blasting is the means of fragmentation in many mining operations and it very important because it affects operational cost directly thus the type of explosive and its properties play a critical role when decision is taken on type of explosive the mine needs. Generally explosives can be classified into two groups primary and secondary. Primary explosive respond to stimuli like shock, impact , friction or flame and pass the state of deflagration to detonation while the secondary explosives can detonate induced by a primary explosives not by deflagration.[3]

Gelatin explosive: Nitroglycerin, is produced by the reaction of gycerin and nitric acid. It is an oily fluid and is sensitive that it can explode due to shock. The use of this type of explosive in the industry requires that it must be absorbed in an inert material or gelatinized. The explosives of this nature are available in three consistencies that are gelatinous, semi-gelatinous and powdery. NG based explosives can be divided into three classes that are dynamites, blasting gelatin and semi gelatin. [3]

ANFO: Ammonium nitrate (AN) was discovered in lease-lend fertilizer disasters at Texas City and Hamburg in the 1940s, however prior to this accidents it was not used in bulk form as a blasting agent. However in the early 1950s AN mixed with fuel oil was introduced to the blast site as a bulk explosive. In many instances the explosive was ideal but it did have its drawbacks. The drawbacks were mainly due to its lack of water resistance and low bulk density, which in turn produced low bulk strength. By the mid-1950, the drilling and blasting was characterised by very high cost of drilling and wet conditions that prompted Cook and Farnam to set out and invent explosives having good water resistance and high bulk strength, over the years modern ANFO was born and is used today as an explosive.[1]

In the commercial explosives the AN percentage varies in the range of 10 – 95% and currently all principal classes of explosive i.e. NG based, dry and wet agents AN is used. When AN is mixed with 5-6% fuel oil, the mixture is known as ANFO. Heavy ANFO is 45 – 50% AN emulsion mixed with prilled ANFO. Loading of ANFO in varying diameter down holes is not complicated because mixed ANFO can be directly poured inside the holes. Small diameter holes favour pneumatic loading because it is quick, compact and thorough. The loading equipment is known as Anoloaders and can exist into two type's pressure and ejector types or combination. [3]

2.3 Case Study: Limestone Quarry in the Philippines

2.3.1 Production Information

The annual production of Quarry was over 3 million tonne of limestone. The geology of the deposit characterised with difficulties owing to frequent shaly and clayey intrusions. The limestone beds, separated at 2-3m interval, were dipping at an inclination of 30^0 to 40^0 towards the pit. The compressive strength of limestone was about 40 MPa, specific gravity of limestone was 2.4. The section of mine comprised of seven benches of 7-9m and consideration was given to physico-mechanical properties and geology of the benches. Two distinctive firing systems (as indicated in Fig.10&11) were designed with staggered blast pattern and the blasts was initiated by shock tube system sequencing of 17ms, 25ms and 42ms. [7]

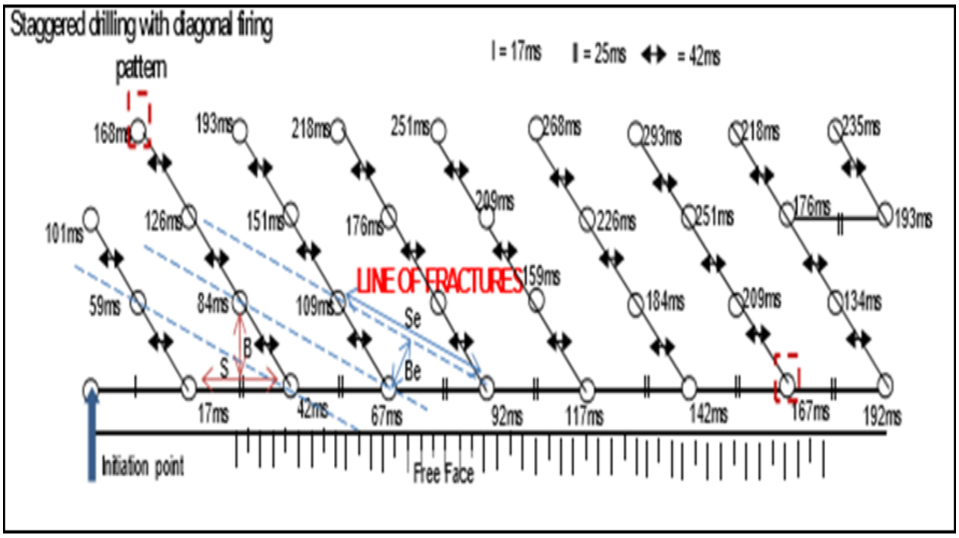


Figure 10: Staggered Drilling with Diagonal Firing System [7]

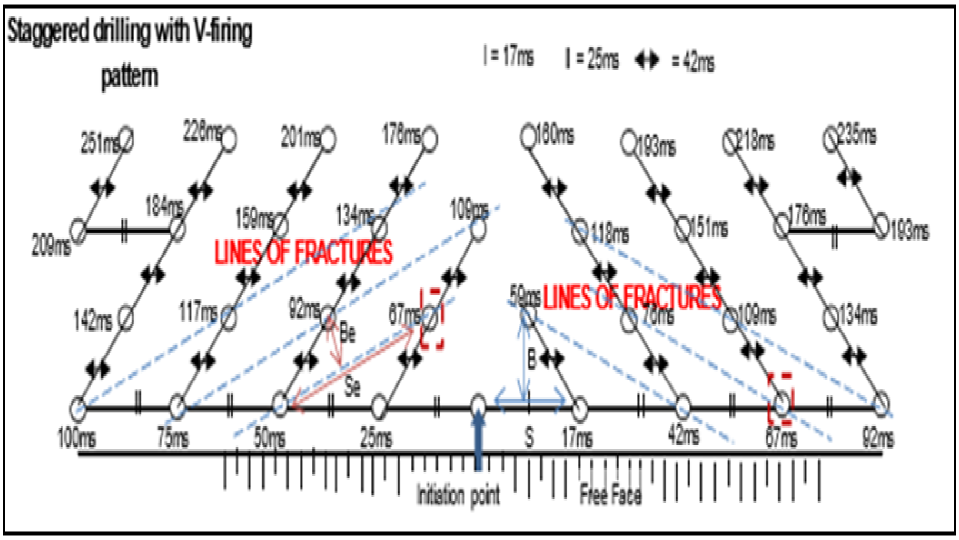


Figure 11: Staggered Drilling with V- Firing System [7]

Explosive used in all the blasts was Ammonium Nitrate Fuel Oil (plant mixed) with shock tube initiation system. The density being 0.8 g/cc and the VOD was 3700 m/s. All the blast rounds were drilled on staggered drilling pattern with ANFO as explosive and sensitized emulsion as primer. The loading equipment used were Front end loader, Shovel and Backhoe and fragmented material was loaded on 35 and 50 tonne truck for hauling. Figure 12 below indicates the longitudinal section of the blast hole. [7]

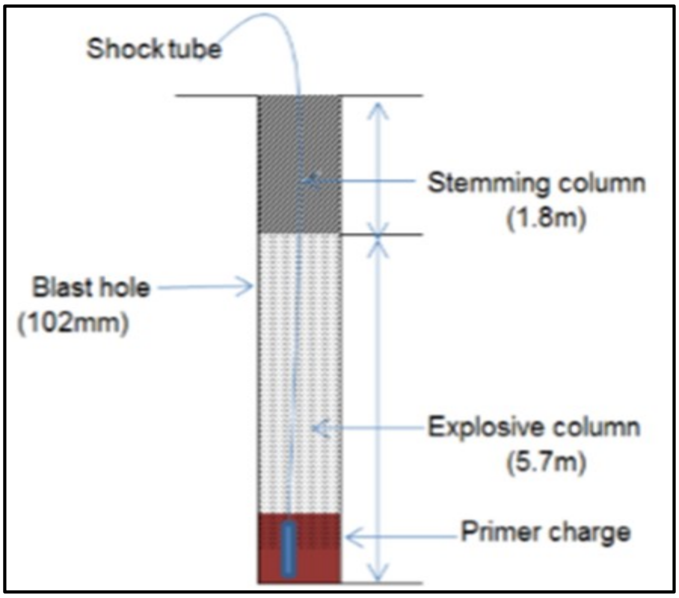


Figure 12: Longitudinal Section of Blast Holes [7]

2.3.2 Production Blast Results

During the comparison of the results from table 2, both firing systems produce good blast in terms of fragmentation. However it was clear from the results that the reduction of mean fraction size (K50) and maximum fraction size(K100) values in V-firing system indicated that the V-firing was better system because it produces uniform material that can make handling much easier for loaders. It is also clear from the results that with the V-firing system parameters such as number of holes increase thus the effect on the powder factor is an increase too and the amount tonnage produce is 2 866tons more. The objective of many mining operations is to increase production at the lowest cost possible and that is provided by the V-firing system in this case study. V- firing systems are far superior to row delays because they produce superior fragmentation due to reduce hole burdens and increased spacing at the time of hole initiation and also due to inflight collision of broken rock during its movement. The delayed action of holes in the back row reduces over break ensuring increased wall stability.

Table 2: Blast Results [7]

Parameters	Diagonal	V-Type
Burden(m)	2.8	2.8
Spacing(m)	3.2	3.2
Depth of holes(m)	6.5	6.5
No of holes	49	69
No of rows	4	5
Total explosive(Kg)	1472	2250
Delay	17/25/42	17/25/42
Throw(m)	8	13
Cycle time(sec)	28.12	29.14
Total tonnage hauled(t)	6134	9000
PF(kg/t)	0.24	0.25
Uniformity, index n	2.44	3.05
Characteristic size Xc	0.34	0.24
MFS K50	0.29	0.21
K100	0.49	0.41

3 PROJECT INVESTIGATION

3. 1 Problem Statement

“The modernization of blasting works in Austin Detonator powder Service Company”

3.2 Objectives

The primary objective of the project is to design a new blast design for the quarry.

Factors under considerations during project to achieve the objectives:

- Current implementation of blasting operations at selected locations of Austin Detonator Power Service.
- Assessment of current blasting parameters in selected location.
- Proposal to change the parameters of blasting if need possible.
- A brief evaluation of technical and economic benefits.

3.3 Methodology

In order to achieve the objectives stated in above section ***(3.2 Objectives)*** the methodology implemented includes Historical, Descriptive, Analytical and Conclusion Oriented Methods.

Historical Method: Information from literature studies was used because it provide better understanding of blast design parameters and their properties. Uncontrollable variables like rock mass characteristics cannot be measured on site but in laboratories thus past literature provides the necessary information that will be significant in modernization of blasting works at the quarry. Past practices at other quarries also contribute to the better understanding of the new blast design. ***(See Chapter 2, 2.3 Case Study).***

Descriptive Method: The modernization of blasting works at the quarry required detail assessment of current practices that are undertaken. To achieve this objective field observation was crucial. A mine visit to Mokra Quarry was contacted to obtain information of the current blasting practice and to use the information obtained in the modernization of the current blasting works. During the field visit important information such as geology, bench geometry, explosives, detonators, blast pattern and general mine information was recorded. The mine workers on site were asked about the logistics and technical aspects of the blast, because their information is vital as they are familiar with the blasting works at the quarry.

Analytical Method: It is impossible to establish a blast design numerically certain empirical rules with equations must be used to enable the blast to be numerically analysed therefor the information obtained from literature studies and through descriptive approach (i.e. field observation) was used in the calculation of new blast design. If the variables can be analysed, optimising of the blast can be done through fine tuning of a production blast's result.

Conclusion Oriented Method: Conceptualize the problem and draw clear understanding between current blast design and proposed new blast design.

3.4 Planning

Project planning is a form of operational planning, whereby the consecutive steps to implement the project activities are mapped out. More important project planning serves as a road map to achieve the objectives set during the course of the project. The modernization of blasting works at Mokra Quarry are no different and did require clear undertaking of what information is available and steps to be taken to achieve desired results. Figure13 below indicates the model that serves as a guideline in project planning and execution, indicating input information, resources, governmental legislation and expected outcome of results.

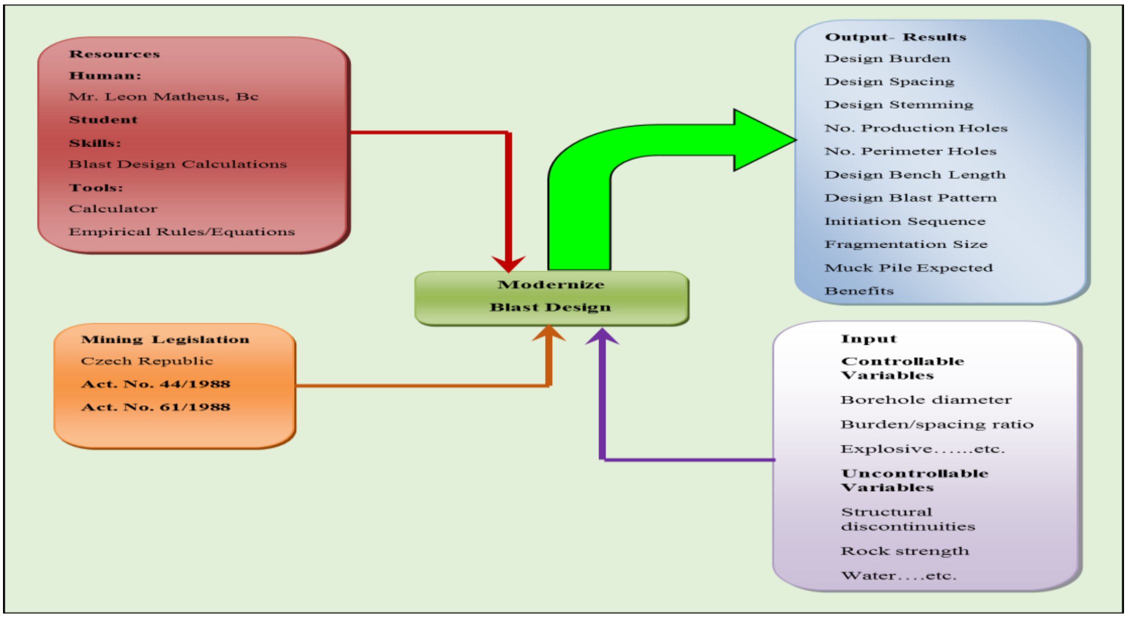


Figure 13: Project Planning Model (author's illustration)

4 CURRENT BLAST DESIGN

4.1 General Information

4.1.1 Bench Information

The information indicated in figure 14 shows data collected in field on a specific bench on which the current blasting work took place. In addition to the information given in figure 14 other information of interest includes borehole diameter of 89mm, burden and spacing approximately 3m respectively, bench length of 100m and bench width of approximately 5m. The stemming material used is either drill remnants or gravel and the two rows were drilled with total number of holes been 48.

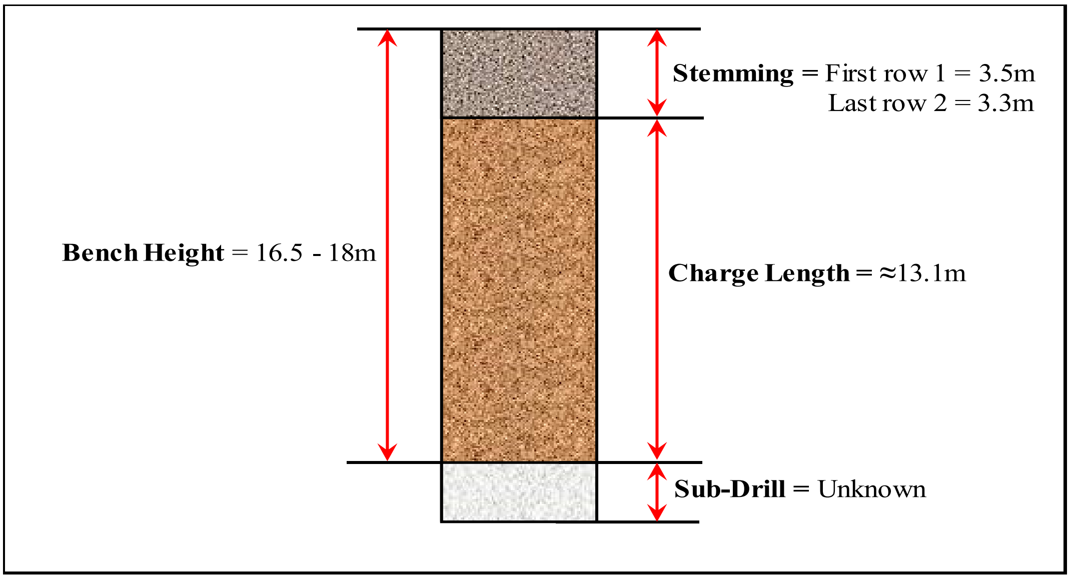


Figure 14: Longitudinal Section of Current Bench Geometry (author's illustration)

4.1.2 Explosive Used

Dynamite Perunit E of **65mm diameter** and ANFO Austinite 3 eco were used on site in the quarry, their specifications discussed below in detail. Dynamite Perunit E (as indicated in Tab. 3) is type of mining explosive, used for rock blasting. It is characterised by high energy content, high detonation velocity and high density. It can be used at mining areas not endangered to explosive potentials such as the risk of explosion of mine gases or air flammable dust mixtures. Other applications are in mine areas where the use of high performance explosives is required to effect material disintegration and underwater blasting's. The explosive is suitable initiator and does not contain carcinogenic DNT and TNT. The important parameters of explosive are summarise in table 3 with picture taken on site at quarry. [28]

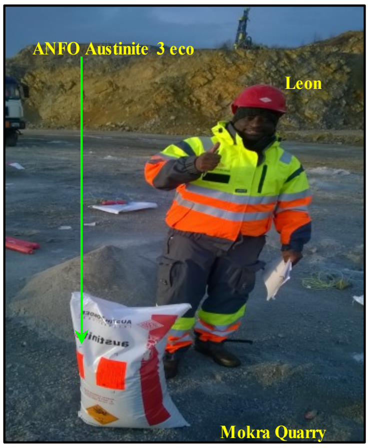
Table 3: Dynamite Perunit E Explosive Specification (author's field photo)

Parameter	**Unit**	**Dynamite Perunit E**	
Type		Plastic	
Explosion heat	$KJ \cdot kg^{-1}$	4100	
Gas volume	$dm^3 \cdot kg^{-1}$	858	
Oxygen balance	% O2	positive 2.2	
Velosity od detonation	$m \cdot s^{-1}$	6000	
Trazul test	ml	385	
Brisance Hess	mm	14	
RWS(blasting gelatine)	%	78	
Density	$Kg \cdot m^{-3}$	1380	
Diameter	**mm**	**65**	
Length	cm	60	
Weight	g	2500	

ANFO Austinite 3 eco (as indicated in table 4) consists of porous ammonium nitrate, mineral oil and aluminium, the presence of aluminium enhances heave. The explosive is not water resistant and thus applicable in dry conditions and is not detonator sensitive. The explosive is not suitable to use in hazardous environments where flammable gases or dust may be found. The advantages include high gas volume, very low sensitivity against mechanical and thermal stress, borehole volume is perfectly utilised thus facilitates high degree of efficiency and may be used for pneumatic loading. [4]

Table 4: ANFO Austinite 3 eco explosive Specification (author's field photo)

Parameter	Unit	ANFO Austinite 3 eco	
Density	$g \cdot cm^{-3}$	0.75	
Explosion heat	$KJ \cdot kg^{-1}$	4050	
Gas volume	$L \cdot kg^{-1}$	937	
Oxygen balance	% O2	0.5	
Velosity of detonation(Confined)	$m \cdot s^{-1}$	3600	
Relative weight strength(ANFO = 100)		90	
Relative bulk strength(ANFO = 0.85 g·cm- 3)		109	



4.1.3 Detonators Used

Non - electric detonators, indicated in figure 15, are used for delays. The non-electric detonators are designed to provide the precise control and accuracy for blasting in surface mines where the risk of ignition of the explosive air – methane or air- coal dust mixtures are not found. Non- electric detonators of 21m length, 475ms with shock tube colour of yellow and non – electric detonator of 6m length, 500ms with shock tube colour of blue are used as in-hole delays. These detonators are designed to be used as down – hole detonators for the initiation of cast boosters, high explosives or pneumatically loaded ANFO. In –hole delays can be equipped with a T- connector for compatibility with detonating cord initiation or can be initiated by another electric or non- electric detonator. [5]

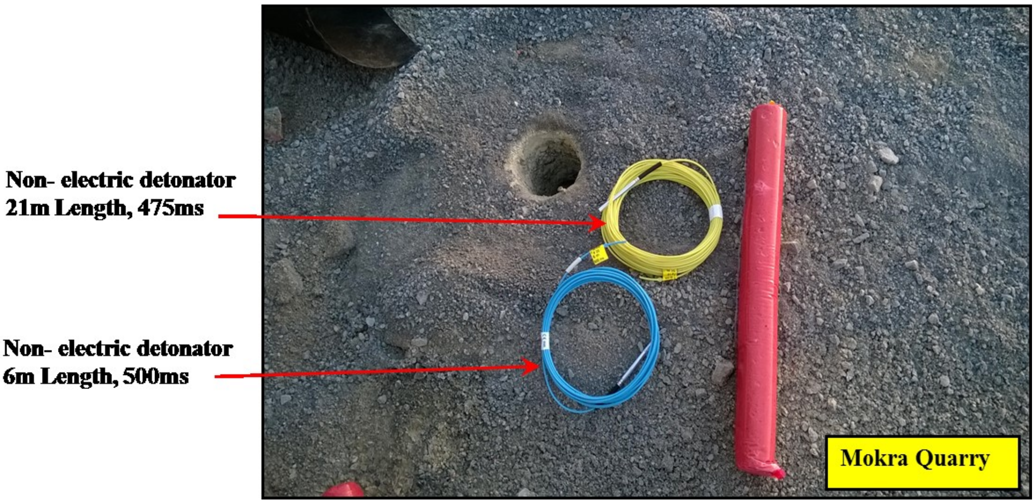


Figure 15: Non –Electric Detonators (author's field photo)

The Shockstar Surface non –electric detonators indicated in figure 16, below are used for inter-hole and inter-row delays. These detonators are available in 9 delays from instantaneous (0ms) to 200ms with shock tube colour of reddish/orange. They are designed for the initiation of other shock tube detonators as a delay between holes and rows. The connector block is designed to enable reliable connections in two positions: **unlocked** position for normal working conditions and **locked** position for most demanding working conditions with a risk of disconnection (e.g. under heavy mats). Surface connectors are equipped with connector blocks able to contain 8 out-going shock tubes yet have a lowered base charge to reduce noise levels and eliminate shrapnel cut-off concerns. Shockstar Surface connections may not be used for initiation of detonating cord. [5]

Figure 16: Non –Electric Detonators-Shockstar Surface (author's field photo)

4.1.4 Blast Design Layout

Current blast design bench layout (as indicated in Fig.17) was drawn with parameters given in **4.1.1.**

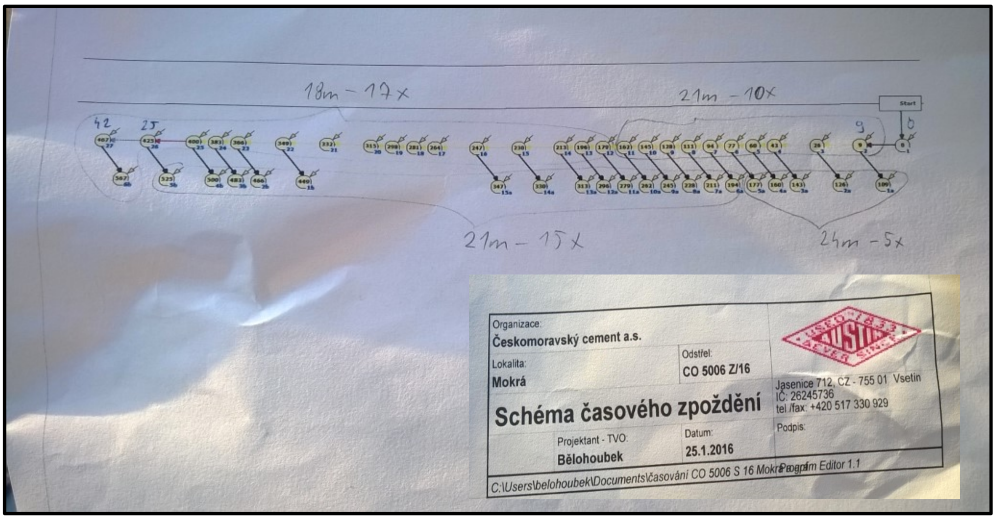


Figure 17: Current Blast Design Layout (author’s field photo)

4.1.5 Economic Benefit

During the study a bulk amount of 250 000CZK was stated by the foreman in field as the total cost that includes drilling, explosives, transport and labour hence no calculation was provided on how the amount was calculated. General salary of employee starting at 900EUR.

4.2 Current Blast Evaluation

4.2.1 Secondary Blast

Perusal of fragmentation results of the bench after the blast indicates presence of un-fragmented segments of rock and uneven face boundary that will result to irregular wall profile after excavation. This can be viewed in detail in figure 18 and figure 19 that indicate photographic evidence and schematic derive from photograph clearly indicating the presence of un-fragmented segments **(1, 3&5)** of rock and uneven face boundary after blast.

This raises the fact that secondary blasting is needed to fragment the segments that are still consolidated. Secondary blasting reduces the large (i.e. primary), blocks to a smaller size but it is expensive and time consuming therefor it slows productivity of any mining operation. Many factors would have contributed to the need of secondary blasting that includes drilling in accuracy, geology, explosive used, blast pattern or the firing system used. The spacing to burden ratio might have also contributed to the existence of un-fragmented segments of rock and uneven face boundary.

In any case spacing provided between two holes in no case should be less than the burden as it causes premature splitting of holes and early loosening of stemming column resulting in sudden drop of blast hole pressure to adversely affect fragmentation. [21]

The other major factor is stemming of the blast holes. Stemming should provide confinement and retention to promote the rock factory by transmitting a major portion of shock as well as gas pressure through the burden rock mass and prior to release of stemming material. Improper confinement results in wastage of this energy leading to poor fragmentation results. [11]

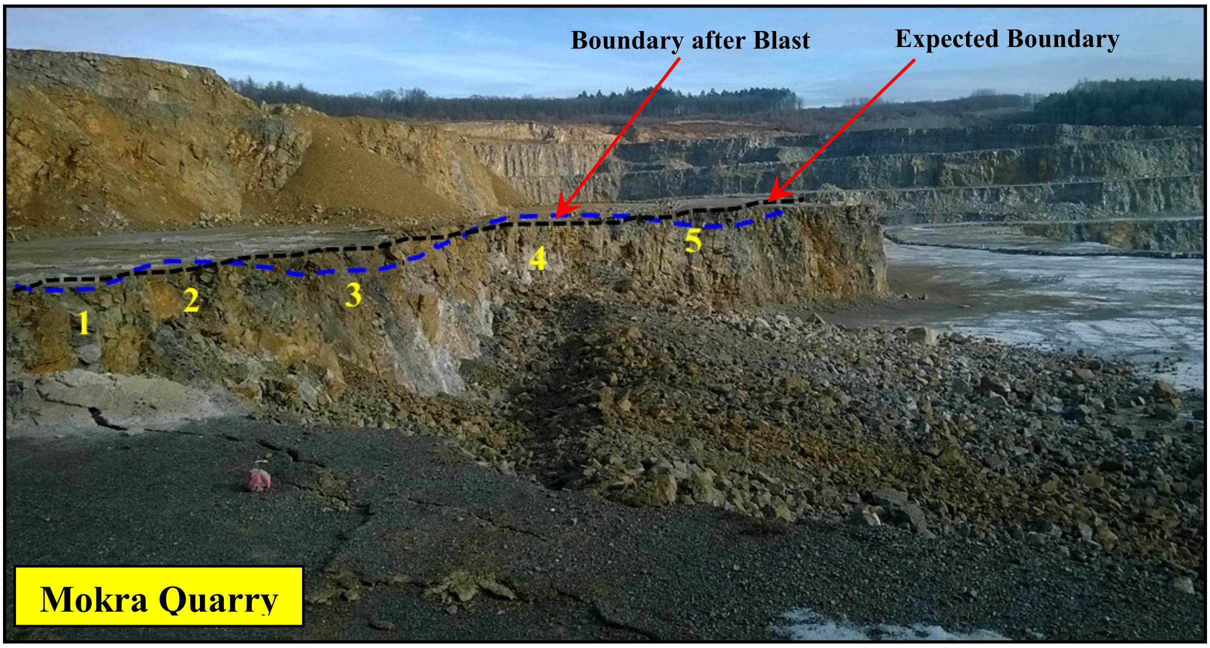


Figure 18: Un-fragmented Segments of Rock (author's field photo)

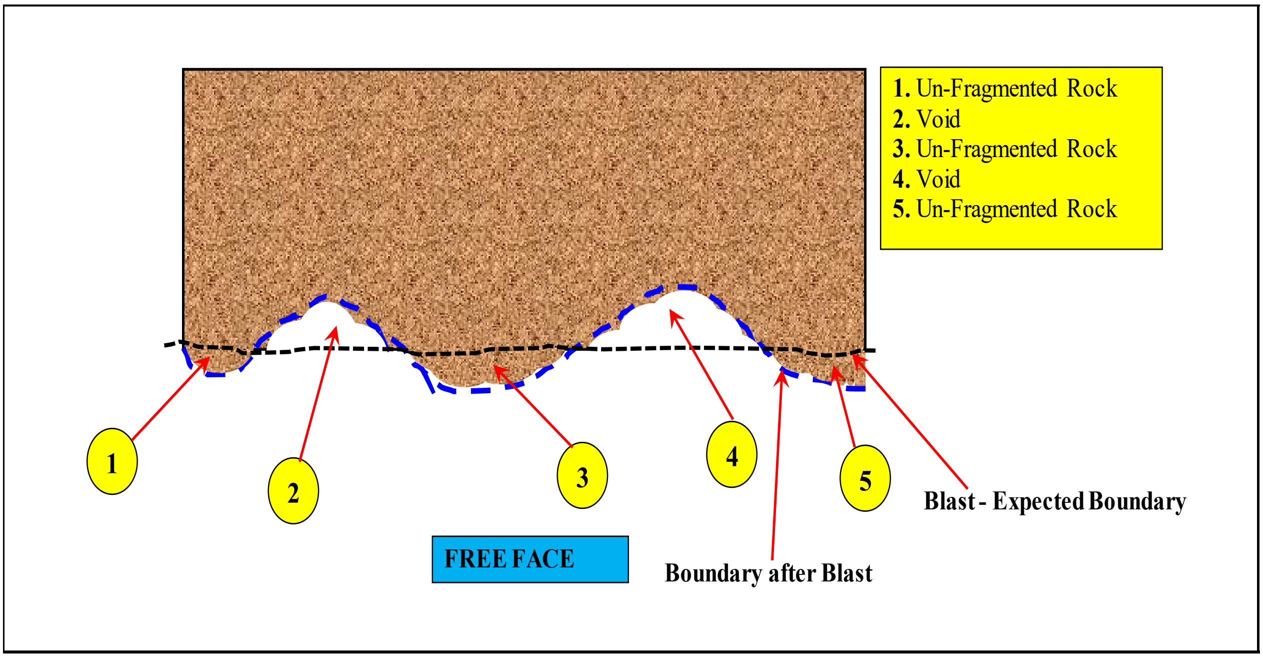


Figure 19: Schematic Un-fragmented Segments of Rock (author's illustration)

In figure 20 below it is clearly visible that blast at Mokra Quarry produced un-fragmented segments of rock and uneven face boundary compared to another quarry in India while both mines used diagonal firing technique. It must be noted that the compared mines might have used the same diagonal firing but the bench geometry or geology is different therefor result of the blast are not the same. However clear distinction can be made between blast that requires secondary blast and the one that does not.

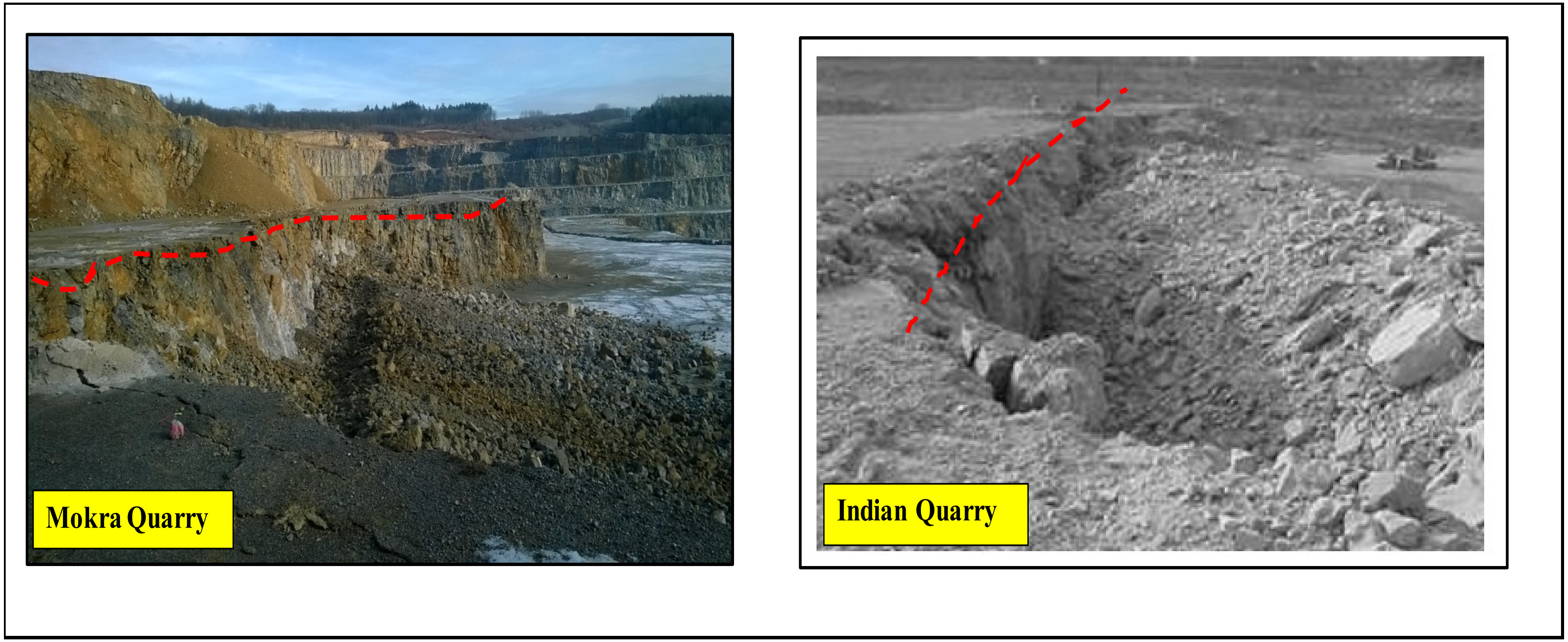


Figure 20: Diagonal Firing Fragmentation Comparison (author’s field photo) and Photo to **right**: [11]

4.2.2 Muckpile Profile

Analysis on Muckpile parameters require understanding of Muckpile shape parameters indicated in figure 21. Throw is the horizontal distance up which center of gravity of blasted material muck lies, drop of Muckpile is the vertically lowering of the blasted muck and lateral spreading is the horizontal distance up to blasted muck lies. [7]

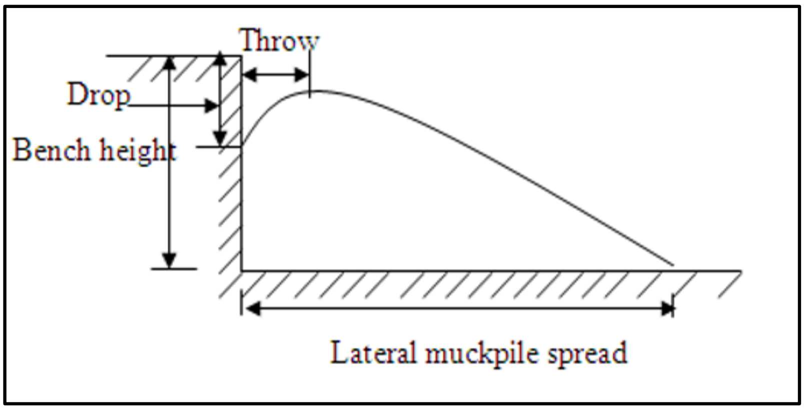


Figure 21: Muckpile Shape Parameters [7]

Analysis on muckpile profile in Mokra quarry in Figure 22 indicates flat profile because of greater throw and drop that has caused an increase in lateral spread. The profile is characterised by large clean up area as material spread, low productivity with rope shovel as loader, high productivity with wheel loader and very safe for equipment operations.

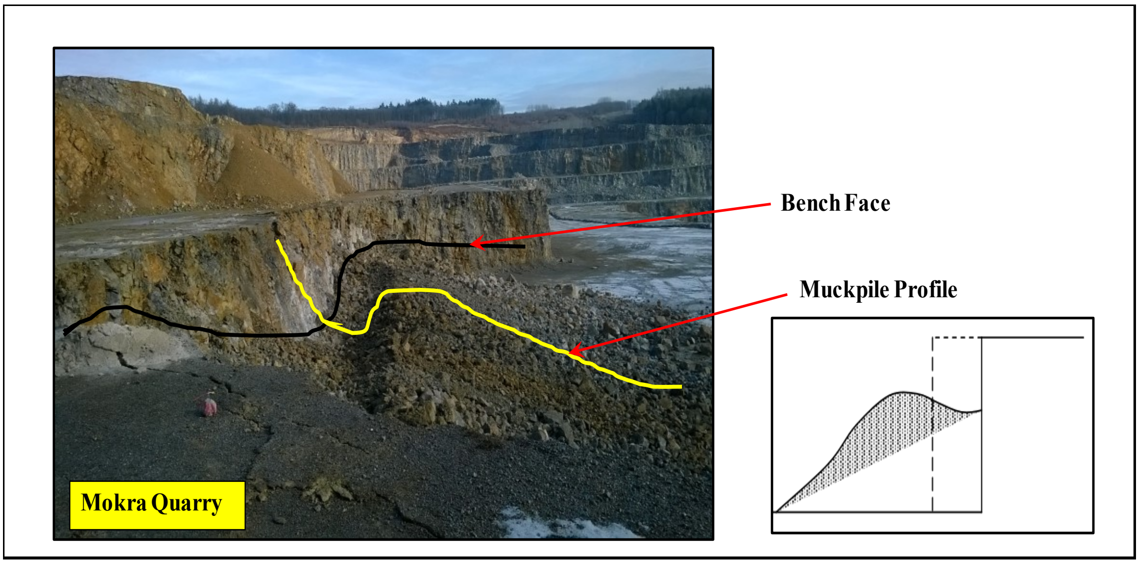


Figure 22: Mokra Quarry Muckpile Profile (author's field photo and illustration)

4.2.3 Fragmentation Size

Fragmentation is used as measuring stick for blast performance. It is the most common driving force in the influence on the blast design. [19]

It is important because poor fragmentation is costly and at times in most quarries fragmentation size is client requirement, therefor the mine is required to provide the client with size of material they desire for their function e.g. aggregate for road construction in civil engineering industry. The two main factors that have the largest effect on fragmentation are, timing and powder factor but geology must also be considered. As indicated in figure 23, below there is possibility that the two factors did play part in the fragmentation size variation of the material.

The material form the blast at Mokra Quarry is not uniform, it has approximately 40% coarse and 60% fines. The coarser fraction been at the bench ends and in distant free face. Fines are found close to and in the middle of the bench face. During the project study there was no information provided from model such as Kuz-Ram model to predict the fragmentation size thus the postulations made in above text is from visual assessment made from photo taken after the blast.

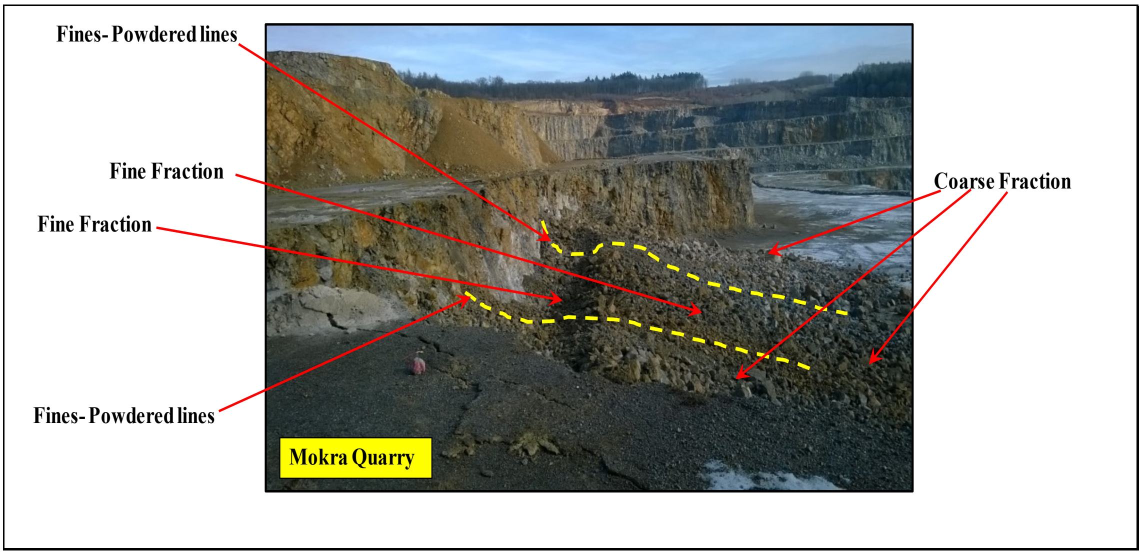


Figure 23: Fragmentation Size Mokra Quarry (author’s field photo)

4.2.4 Blast Design Logistics

A firing pattern is like an electrical circuit providing a pathway for detonation wave in providing that explosive charges in the blast holes can initiate. To this end the firing system and pattern determines the movement and direction of the rock by creating free face for subsequent blast holes/rows. [21]

Mokra Quarry uses a diagonal firing pattern as indicated in figure 24. Two Dynamite Perunit E explosive of 60cm in length with weight of 2500g are placed in bottom of drilled hole and one filled with non-electric detonator of 21m length, 475ms with shock tube colour of yellow. One Dynamite Perunit E explosive of 30cm in length with weight of 1250g is placed at top of drilled hole and filled with non-electric detonator of 6m length, 500ms with shock tube colour of blue. The fact that two non-electric detonators are used is because Czech law that states if the hole depth exceeds 12m, it is required to have two detonators. Then ANFO Austinite 3 eco is loaded and stemming material either gravel or drill remnants are added to seal the hole and in-hole delays are connected to Shockstar Surface non –electric detonators with instantaneous (0ms) to 200ms with shock tube colour of reddish/orange.

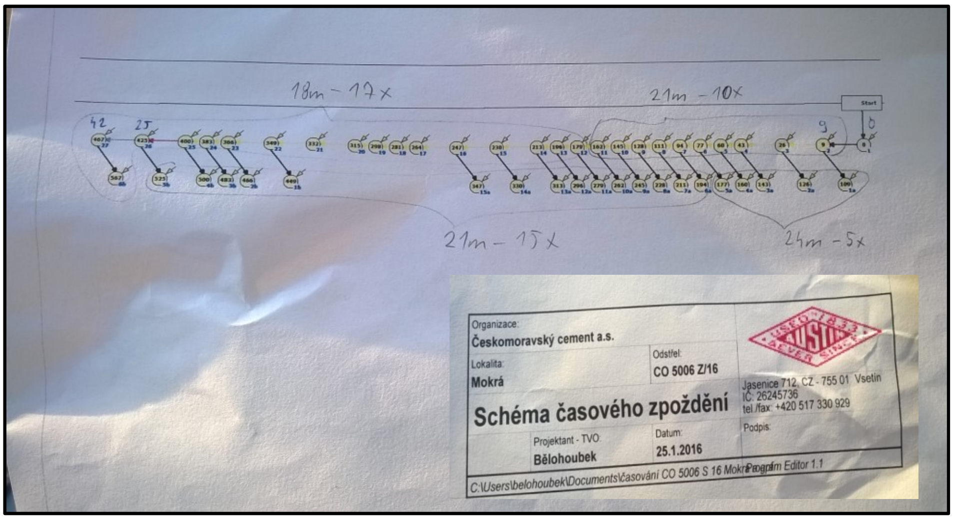


Figure 24: Diagonal Firing Pattern (author's field photo)

5 PROPOSED BLAST DESIGN

5.1. Design Information

5.1.1 Bench Information

- Bench Height = 16.5 – 18m, Bench Length = 100m, Bench Width = 5m
- Burden ≈ 3m, Spacing ≈ 3m
- Charge Length ≈ 13.1m
- Stemming First row = 3.5m and Last row = 3.3m
- Sub - Drill unknown
- Number holes in total = 48
- Borehole Diameter = 89mm

5.1.2 Rock Properties

Information with regard to rock properties was to my own discretion and does not necessary provide the true values of Mokra Quarry. The actual information on rock properties of the quarry was not made available to me.

Limestone Rock;

- Friction angle of rock joint = 30 - 40^0 [8]
- Density = 2646Kg/m^3
- Compressive strength of rock = 50 - 150MPa
- Angle of roughness of rock joint = 5^0 [8]
- Rock Quality Designation = 50 - 75%, Description of rock quality been fair. [15]

Clay Rock;

- Friction angle of rock joint = 20 - 35^0 [8]
- Density = 1900Kg/m^3
- Compressive strength of rock = 4- 12MPa
- Angle of roughness of rock joint = 5^0 [8]
- Rock Quality Designation = 50 - 75%, Description of rock quality been fair. [15]

5.1.3 Explosives and Specifications

Dynamite Perunit E explosive of **70mm diameter** (as indicated in Tab.5) is the explosive suggested for use in the proposed blast design. The explosive is used for rock blasting in quarry and is characterised by high energy content, high detonation velocity and high density.

Table 5: Design Dynamite Perunit E Specifications [28]

Parameter	Unit	Dynamite Perunit E
Type		Plastic
Explosion heat	$KJ \cdot kg^{-1}$	4100
Gas volume	$dm^3 \cdot kg^{-1}$	858
Oxygen balance	% O2	positive 2.2
Velosity of detonation	$m \cdot s^{-1}$	6000
Trazul test	ml	385
Brisance Hess	mm	14
RWS(blasting gelatine)	%	78
Density	$Kg \cdot m^{-3}$	1380
Diameter	**mm**	**70**
Length	cm	60
Weight	g	2500
Water resistance		Very good
Shelf life	Month	9

ANFO Austinite 3 eco (as indicated in Tab.6) is the explosive suggested for the use in the proposed blast design. The explosive is used for rock blasting in quarry and it consists of porous ammonium nitrate, mineral oil and aluminium, the presence of aluminium enhances heave. The explosive is not water resistant and thus applicable in dry conditions and is not detonator sensitive. [4]

Table 6: Design ANFO Austinite 3 eco Specifications [4]

Parameter	Unit	ANFO Austinite 3 eco
Density	$g \cdot cm^{-3}$	0.75
Explosion heat	$KJ \cdot kg^{-1}$	4050
Gas volume	$L \cdot kg^{-1}$	937
Oxygen balance	% O2	0.5
Velosity of detonation(Confined)	$m \cdot s^{-1}$	3600
Relative weight strength(ANFO = 100)		90
Relative bulk strength(ANFO = 0.85 g·cm-3)		109

5.2 Basic Blast Design Calculations

5.2.1 Equations

Blast design requires iterative approach as some variables are inter-related, how many iteration are required depends on the balance of the design. It is impossible to establish a blast design numerically, certain empirical rules with equations must be used to enable the blast to be numerically analysed. Equation 1- 4 will form the basis of the proposed blast design calculation. First the mass of explosive per metre is calculated using equation 1.Since equation 1incorporates coupling then equation 2 can be used to calculate coupling. Secondly a powder factor (PDF) must be calculated using equation 3. If the calculated powder factor (PDF) is suitable a recommended burden is calculated using equation 4. [14]

Although equations are used there is no standard formula to calculate blast design, the final blast design is a product of adjusting controllable variables to meet a good or balance blast design.

$$M_c = \frac{\pi . d^2 . \rho_w . c}{4} \qquad \textbf{[1]}$$

Where: M_C = Mass of explosive per linear metre

d = Drill hole diameter in metres

$\dot{\rho}_w$ = Explosive density

c = Coupling factor (1 for poured or pumped explosives)

$$c = \left[\frac{d_e}{d}\right]^{2,4} \qquad \textbf{[2]}$$

Where: d_e = Explosive diameter in metres

d = Drill hole diameter in metres

$$PDF = 94.6 \log \left[\frac{3{,}3\rho . Tan(\phi + i) . \sqrt[3]{\sigma_c} \left(\frac{d}{100} \right)^2}{115 - RQD} \right] + 540 \qquad [3]$$

Where: PDF = Powder factor (Kg/BCM)

$\dot{\rho}$ = Rock density (t/m^3)

φ = Friction angle of rock joint (^{0})

i = Angle of roughness of rock joint (^{0})

d = Drill hole diameter (mm)

σ_c = Compressive strength of rock (MPa)

RQD = Rock quality designated according to Deere's classification (%)

$$B = \frac{-(A.M_c.Y) + \sqrt{(A.M_c.Y)^2 + 4(H^2.M_c.A.PDF)}}{2(H.PDF)} \qquad [4]$$

Where: A = Burden to spacing ratio

Y = Stemming to burden ratio

H = Bench height (m)

B = Burden (m)

5.2.2 Calculated Results

The results in Table 7, below were calculated using the ***equations in 5.2.1***

Table 7: Proposed Blast Design Results

Parameter	**Units**	**Results**
PDF	0.62 $Kg{\cdot}m^{-3}$	**Balance Blast - Empirical Rules**
C	0.3	
M_c	6.9 $Kg{\cdot}m^{-1}$	
B_{design}	2.8m	

With the calculated results in mind I postulate the following;

S = 1.2B Guarantee that the charge breaks out in the required direction.

T = 1.1B Estimate starting point to avoid the charge breaking through to the surface causing fly rock.

5.2.3 Blast Design Dimensions

- Standard deviation taken as 1.2
- Burden (B) = **2.8m**
- Spacing(S) = 1.2B = (1.2) * (2.8) = **3.4m**
- Stemming (T) = 1.1B = (1.1) * (2.8) = **3.1m**
- Sub-Drill (U) = 0.5B = (0.5) * (2.8) = **1.4m**
- Bench Height (H) = **18m**
- With Equation H = T + L + U
- Charge Length (L) = **13.5m**
- Bench Length = **60m**
- Bench Width = **16m**

Figure 25 below indicates schematic of the proposed blast design bench geometry.

Bench Geometry – Not drawn to scale;

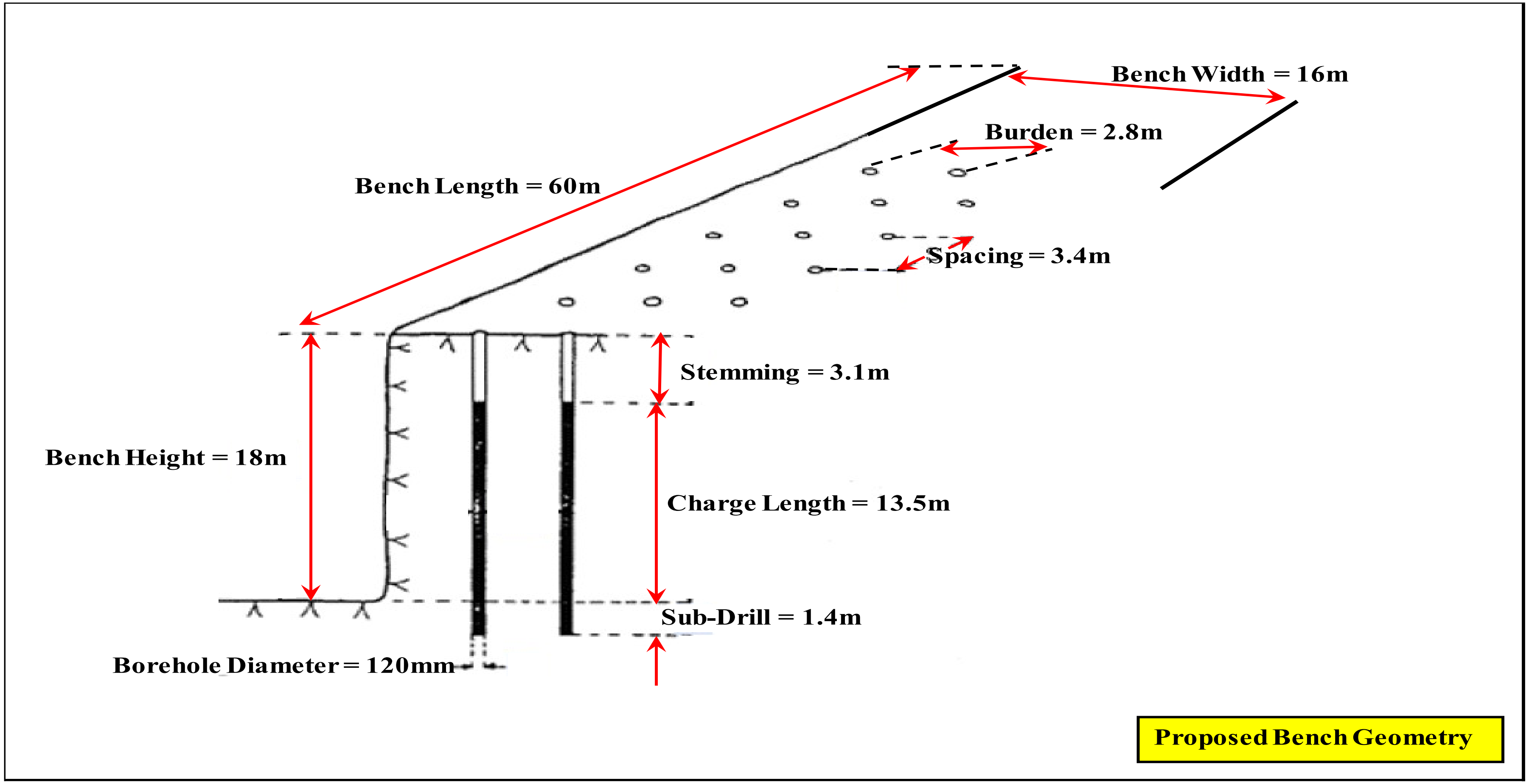


Figure 25: Proposed Bench Geometry (author's illustration)

5.3 Production

5.3.1 Specified Parameters:

Average rock density = 2273Kg/m^3

Bench Width ≈ 16m

Bench Thickness = 18m

Production Capacity = 5000t/day

Losses = 12%

Planned Production Tonnage **(per week)** = [5000tons/day * 7days/week] + 12% for losses

= 39 200t/week

Volume = [39 200t/week] / [2.273t/m^3] = **17 245.93m^3**

Design Bench Length = [17 245.93m^3] / [16m * 18m] = 59.8 ≈ **60m**

5.3.2 Blast Hole Numbers

Consideration was made for control blast thus burden and spacing were adjusted for the back row. The new dimension of spacing being 1.07 of design burden and the burden 0.5 of design burden and then the number of holes for design bench were calculated.

Number of holes (Production Blast) **=** ([60-3.4]/3.4) * ([16-2.8]/2.8) **= 85 holes**

Number of holes (Perimeter Blast) **=** ([60-3]/3) = **19 holes**

Total Number of holes = [85+19] **= 104holes**

5.3.3 Blast Pattern and Firing System

Square blasting pattern with closed chevron V1 firing system was proposed for the new blast design and is indicated in figures 27, 28, 29, and 30. The blast hole configuration for the proposed blast design is indicated in figure 26, with in-hole delay of 500ms (top) and 475ms (bottom), the inter-hole delay and the inter-row delay will be 100ms.

The fact that the hole has two non-electric detonators is because of Czech mining law (Act. No. 61/1988) that states if blast hole depth exceeds 12m, it is required to have two detonators [26]

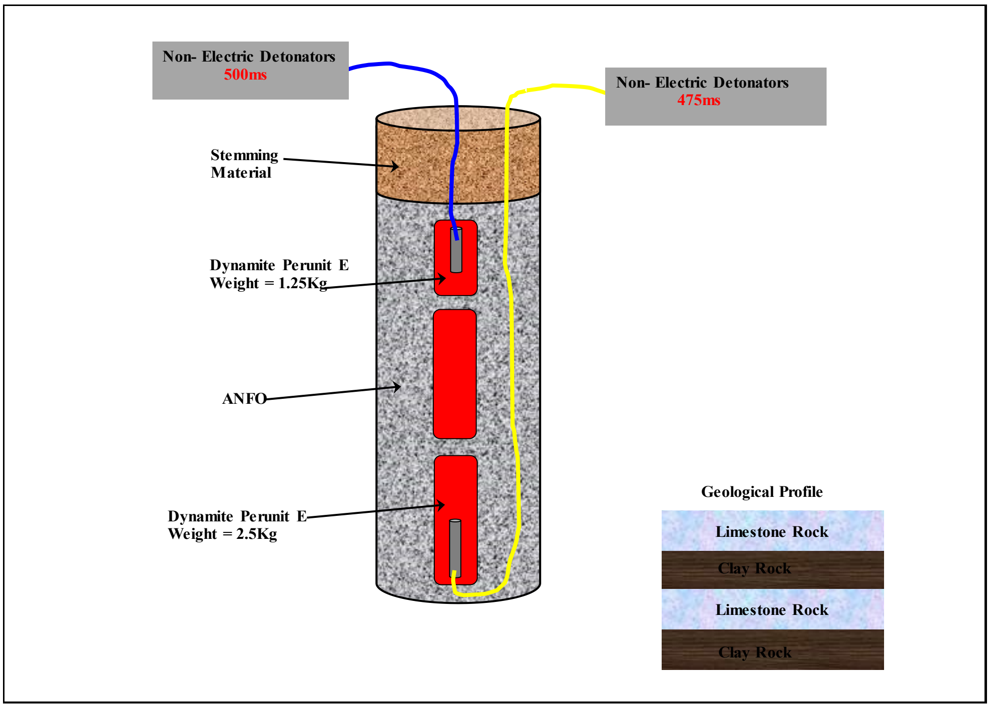


Figure 26: Blast Hole Configuration (author's illustration)

Proposed Drill Hole Layout – Not drawn to scale;

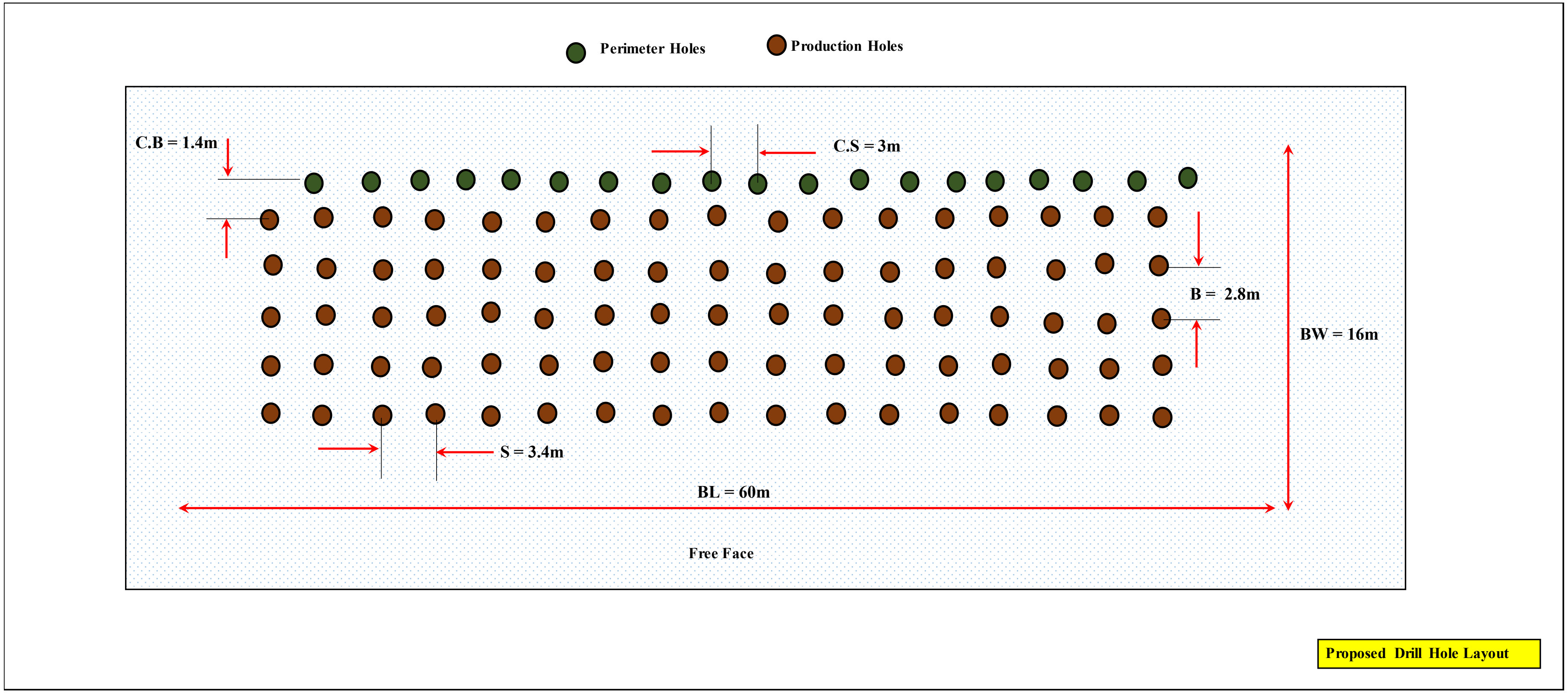


Figure 27: Proposed Drill Hole Layout (author's illustration)

Proposed Section of Production Holes – Not drawn to scale;

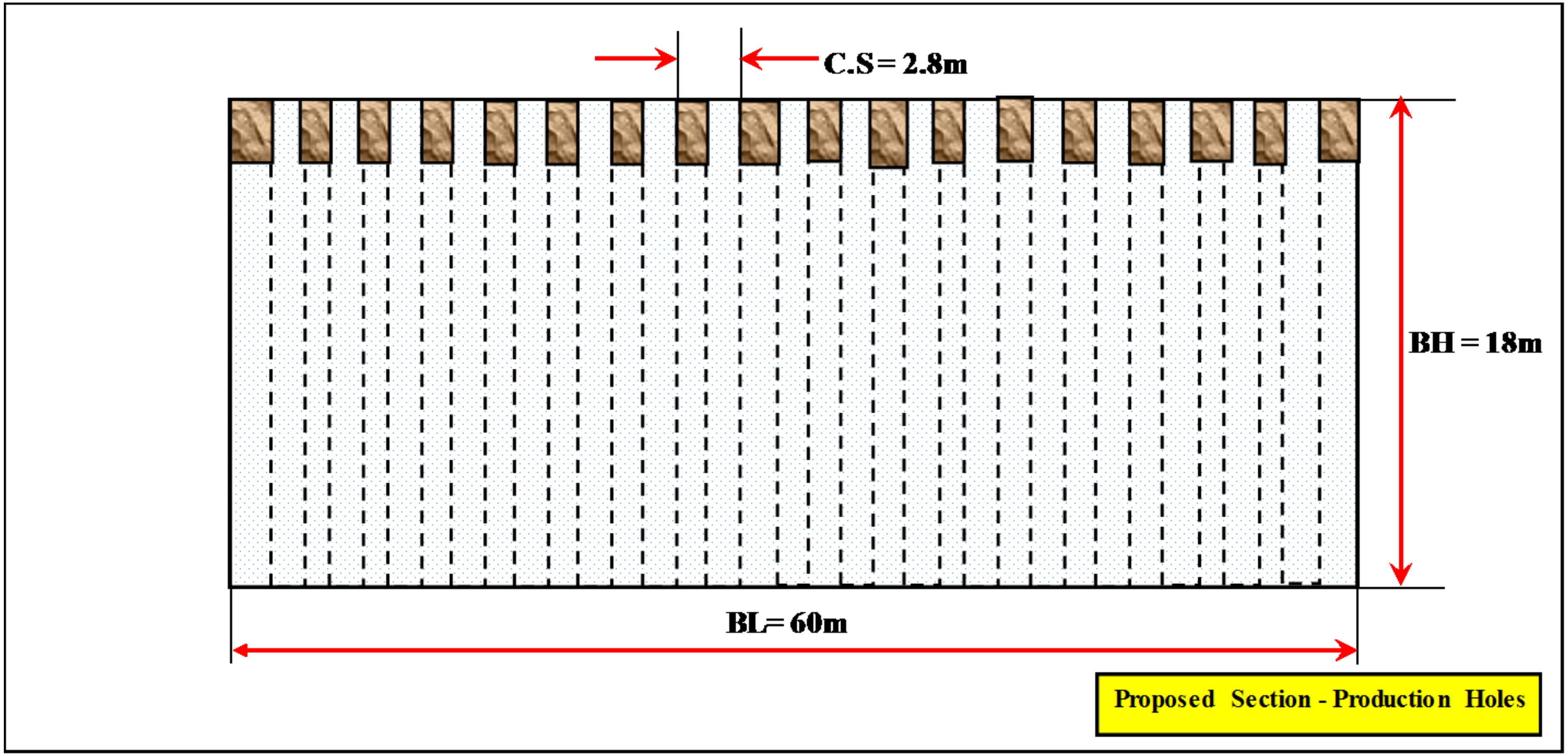


Figure 28: Proposed Section Production Holes (author's illustration)

Proposed Section of Perimeter Holes- Not drawn to scale;

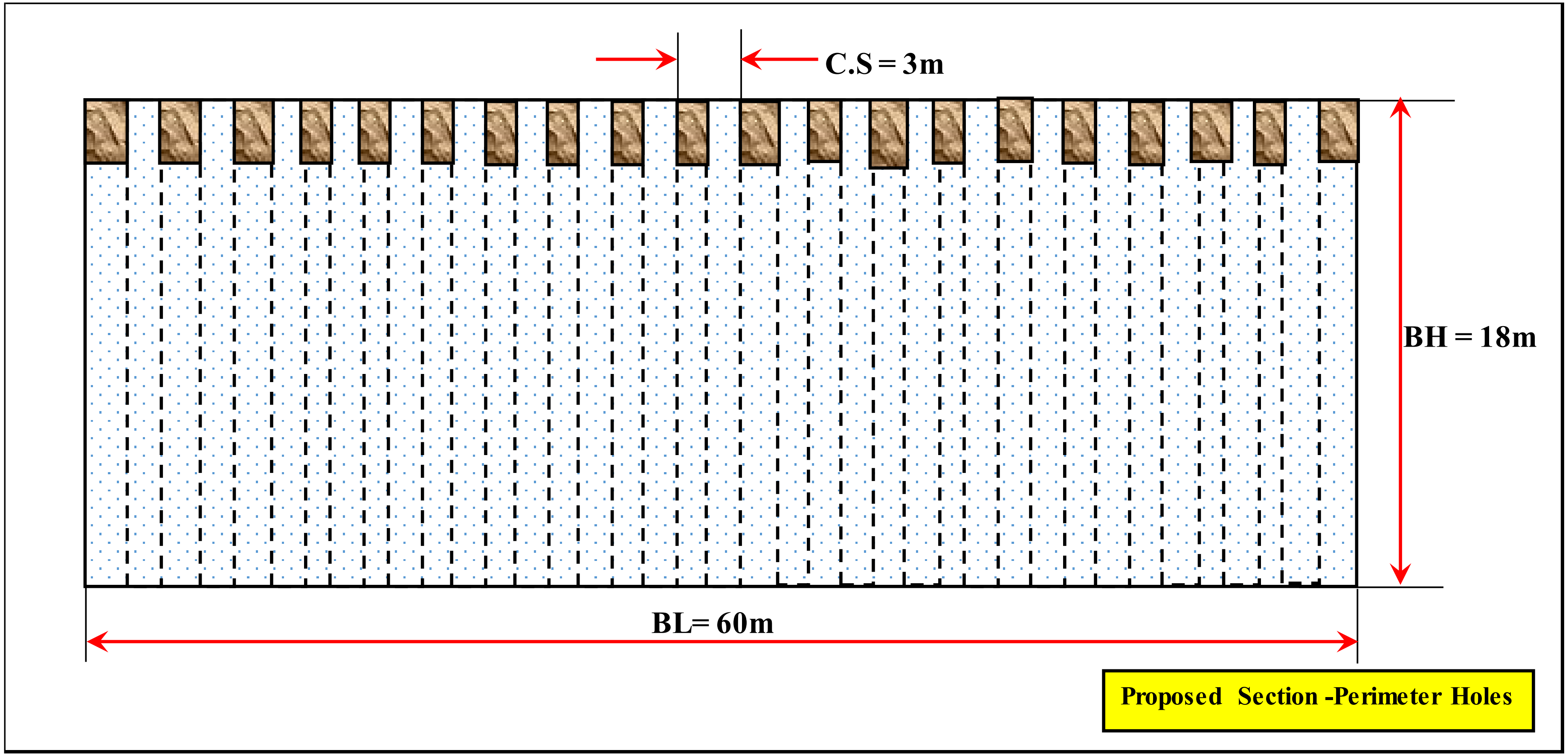


Figure 29: Proposed Section Perimeter Holes (author's illustration)

Proposed Firing Sequence – Not drawn to scale;

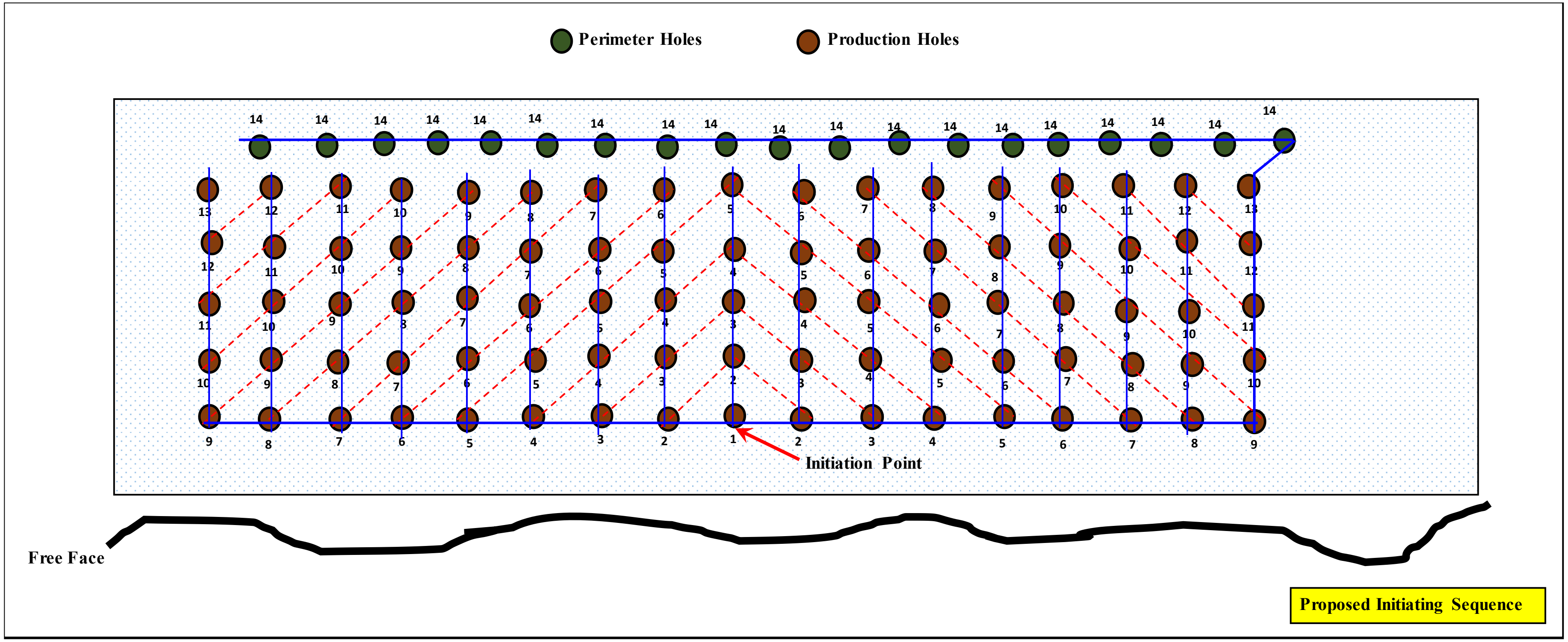


Figure 30: Proposed Firing Sequence (author's illustration)

5.4 FRAGMENTATION

5.4.1 Kuz-Ram Model

Kuz - Ram Model was used as the basis to predict the fragmentation expected from the proposed blast design. Kuz-Ram model is fundamentally a set of three equations used as function of another and used to calculate the characteristic size of blasted material. The three set of equations are Kuznetsov equation, Rossin-Rammler exponent and Rossin-Rammler equation. Blast parameters such as rock properties, explosive specifications, design burden, spacing and charge length are used in the equations. In addition weight of explosive per hole, standard deviation of drilling spacing, relative weight strength of explosive and rock factor are used in the equations. [14]

5.4.2 Blast Design Fragmentation

Equation 1: Kuznetsov equation is used to calculate average fragment size of material using blast parameters with rock factor of $A_f = 7$ because of the well-defined closely spaced weak joints.

$$x_{avg} = A_f \left[\frac{M_h}{Vo}\right]^{-0.8} M_h^{0.167} \left[\frac{RWS}{115}\right]^{-0.633} \approx \mathbf{23.036014 = 23.0\ cm}$$

Where: X_{avg} = Average fragment size (cm)

A_f = Rock factor

M_h = Mass of explosive per hole (kg)

V_o = Volume blasted per hole (m^3)

RWS = Relative weight strength of explosive (relative to ANFO as 100)

Equation 2: Rossin - Rammler equation, with R as mass fraction of fragmentation larger than X(cm) and n is the uniformity index usually between 0.8 - 2.0.

$$R = exp^{-\left(\frac{x}{xc}\right)^n}$$

Where: R = Mass fraction of fragments larger than size x

X = Fragment size (cm)

n = Rossin- Rambler exponent (constant)

X_C = Characteristic fragment size (constant)

Equation 3: Rossin - Rammler exponent is calculated using blast parameters with standard deviation in spacing of ω = 1.2 used.

$$n = \left(2{,}2 - 14.\frac{B}{d_e}\right).\left(1 - \frac{\omega}{B}\right).\left(1 + \frac{\left(\frac{1}{A} - 1\right)}{2}\right).\frac{L}{H} \approx \mathbf{0.8530953 = 0.9}$$

Where: L = Charge length

Taking into consideration the three equation mentioned above and with average size of 23.0cm and n of 0.9 when R = 50% using Kuznetsov equation in the Rossin – Rammler equation the characteristic size(cm) of the material was calculated.

When R = 0.5 **Rossin – Rammler equation written as;**

$$R = exp^{-\left(\frac{x}{xc}\right)^n}$$

Then $X_c = [X_{avg}] / [0.693]^{1/n} = [23.0]/[0.693]^{1/0.9} =$ **35.4cm**

5.4.3 Size Distribution Curves

Analysis of size distribution curve of percent product **passing** (as indicated in Fig.31) of the proposed blast design constructed from Kuz-Ram model the K50 value which represents the screen size through which 50% of the loosened rock would pass if screen is 23(K50 = 23cm), the size distribution curve indicates low K50 value that can represent material of optimum size not necessary fines.

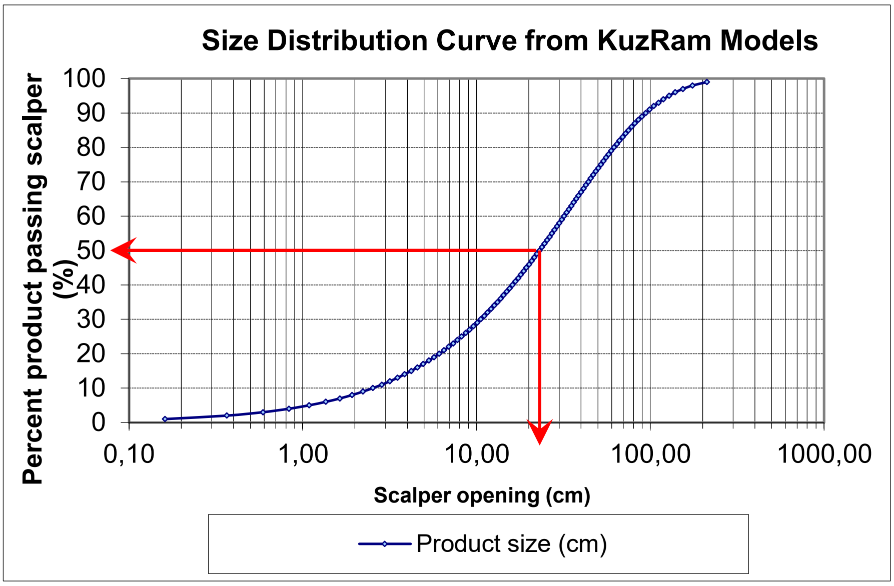


Figure 31: Size Distribution Curve – Percent Passing

Analysis of size distribution curve of percent product **retained** (as indicated in Fig.32) of the proposed blast design constructed from Kuz-Ram model it can be deduced from the curve that at open setting of 90cm on scalper, 10% of the product will be retained. Considering Mokra Quarry has Gyratory and Hammer crusher scalping is of less importance because Gyratory crushers are not sensitive to fines and handle the excavated material (ore) as such. The mechanics of Gyratory crusher of large, steep with relative short stroke compared to settings that allow fines to flow through more easily.

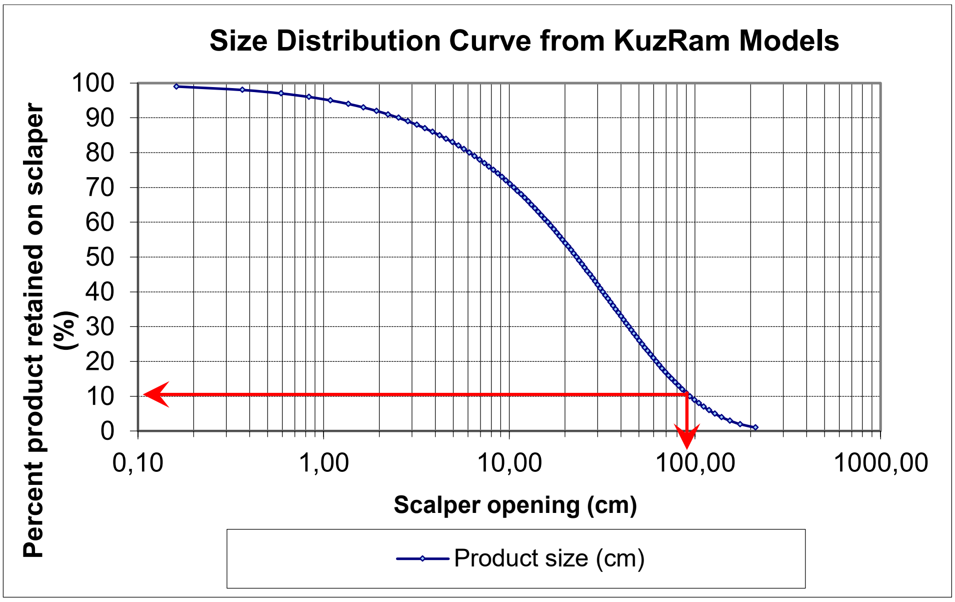


Figure 32: Size Distribution Curve – Percent Retained

5.4.4 Muckpile Profile

The shape of the fragmented rock pile can be controlled by the firing sequence to achieve certain type of results to suit the loader requirement. In the proposed blast design square blasting pattern with closed chevron V1 firing system was chosen. The closed chevron firing pattern rips out V shape wedge and tends to throw material into centrally collected muck pile best suited for hydraulic shovel or large wheel loader which can dig high faces. [25]

The centrally collected muckpile forms because of collisions of ejected burdens from both sides of the chevron that leads to a loss in momentum in the moving rock and displacement is thus less. Figure 31 below indicates the predicted muck pile for the design blast.

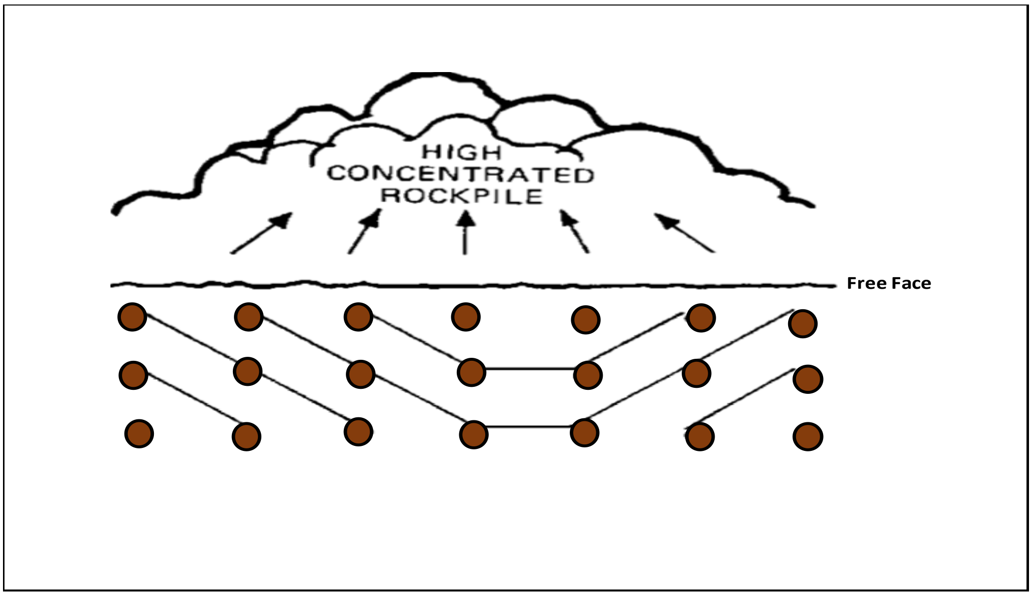


Figure 33: Proposed Design Muckpile Profile [14]

6 COMPARISON AND EVALUATION OF BLAST DESIGNS

6.1 Blast Parameters

Blast design is an important aspect of modern day quarry mining operations. The need of design is important not only to satisfy client requirement but also to minimize cost during production at the mine. Design is crucial because it affects fragmentation of rock and it gives the mine ability to predict the material size thus can decide what is best suited for their loading, hauling and their crusher equipment. Table 8 below indicates information from current blast design and proposed blast design. For the purpose of this study only pattern/firing system, fragmentation size, muckpile profile, secondary blasting and benefits will be compared.

Table 8: Blast Parameters Comparison

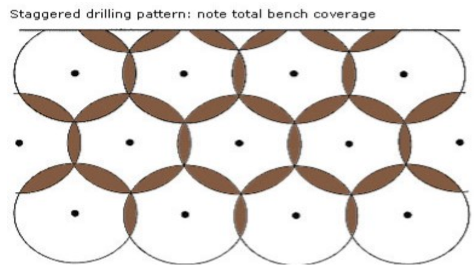
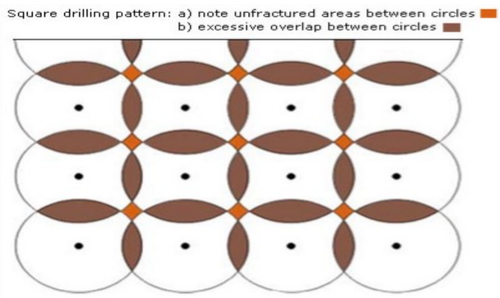
	Blast Design	
Parameters	**Current**	**Proposed**
Borehole diamter(m)	0.089	0.12
Explosive diameter(m)	0.06	0.07
Bench height(m)	16.5-18	18
Burden(m)	3.0	2.8
Spacing(m)	3.0	3.4
Stemming(m)	3.4	3.1
Charge length(m)	13.1	13.5
Sub-drill(m)	unknown	1.4
PDF($Kg \cdot m^{-3}$)	unknown	0.62
No. of holes	48	104
No. of rows	2	5
Delays(ms)	same	same
Firing pattern	Staggered	Square
Initiation system	Diagonal	Chevron V1
Uniformity index, n	unknown	0.9
Characteristic size Xc (m)	unknown	0.35
K50(m)	unknown	0.23

6.1.1 Pattern and Firing System

6.1.1.1 Pattern

The choice of the blast pattern is not random but chosen based on effectiveness and experience of the past. Table 9 below indicates the general characteristics of the two main patterns used not the outcomes reflected after the blast. The current blast design has S=B, with spacing to burden ratio of 1, thus the expected cover is 98.5%. Hence the proposed design has S>B, with spacing to burden ratio of 1.2, thus the expected cover of 99.7% if proposed blast is fired on staggered pattern. [14] [31]

Table 9: Blast Pattern Characteristics [14]

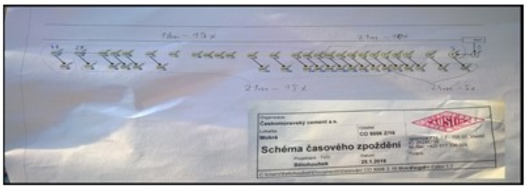
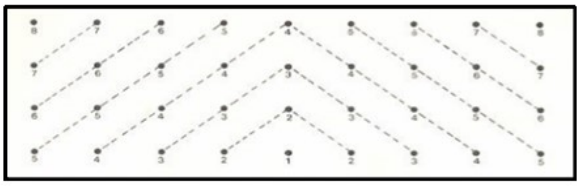
	Blast Pattern	
Current	**Staggered Pattern** 1. Uniform distribution 2. Easy to position holes incorrectly 3. Good/Even fragmentation **(S>B)**	
Proposed	**Square Pattern** 1. Fair distribution 2. Easy to position holes correctly 3. Good fragmentation **(S>B)**	

6.1.1.2 Firing System

Sequential and proper relief to the successive burden rock mass is an essential pre-requisite for the success of any blasting program. To this end, the blast pattern with the firing system decides the movement and direction of rock by creating free face for subsequent holes/rows. [17] [23]

Each firing system with pattern used has its own application and is selected on the basis of performance. Proper use of firing system with pattern can produce optimal blast performance in terms of fragmentation and cost associated with loading and hauling. The current blast design uses staggered drill pattern with diagonal initiation system and the proposed blast design uses square drill pattern with closed V1 chevron initiation system. It is anticipated that the proposed design will be drilled accurate. The delay in current and proposed blast design is the similar with in-hole delays of 500ms (top) and 475ms (bottom), the inter-hole delay and the inter-row delay will be between 100ms. The fact that the hole has two non-electric detonators is because of Czech mining law ***(Act. No. 61/1988)*** that states if a hole depth exceeds 12m, it is required to have two detonators. The results of the current and proposed blast design are indicated in table 10. The fragmentation of the proposed blast design is merely a hypothesis as during the study the proposed blast was not executed, however table gives clear view of expected fragmentation outcomes.

Table 10: Blast Fragmentation Results (author's field photo) Photo to right [7]

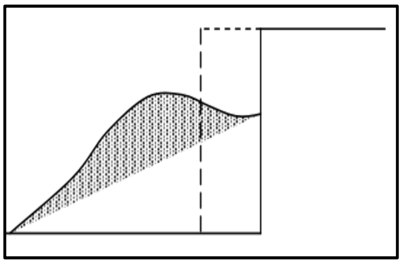
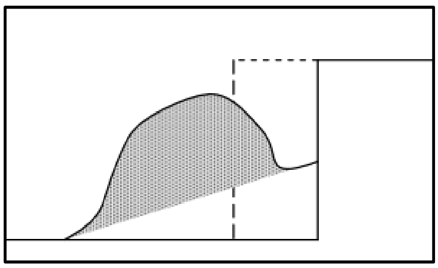
Current - Blast Results	Proposed - Blast Results
	
	

6.1.2 Muckpile Profile

In the proposed blast design the material is less spread and does not require a dozer for loading preparation as in current blast design. The use of loader or dozer to prepare material for loading is costly thus in new blast design the loader is more concentrated on loading. The availability of large free and the chosen firing system for the proposed blast design allows minimum throw and drop and the fragmented material height will not exceed initial bench height dimensions and therefor can be loaded with larger wheel loader or hydraulic shovel. Table 11, below indicates the schematics and brief summary with regard to muckpile characteristics of the two blast. The closed chevron firing system rips out V shape wedge and tends to throw material into centrally collected muckpile best suited for hydraulic shovel or large wheel loader which can dig high faces. [25]

The centrally collected muckpile profile forming in proposed blast design muckpile profile is due to collisions of ejected burdens from both sides of the chevron that leads to a loss in momentum in the moving rock and displacement is thus less.

Table 11: Muckpile Profile Comparison (author's illustration)

Current Blast Design Muckpile Profile	**Proposed Blast Design Muckpile Profile**
	
1. Large clean up area as material is spread	1. Low clean up area as less material is spread
2. Low productivity with hydraulic shovel as loader	2. High productivity with hydraulic shovel as loader
3. High productivity with wheel loader	3. High productivity with large type of wheel loader
4. Very safe for equipment operation	4. Safe for equipment operation

6.1.3 Fragmentation Size

Fragmentation from current blast design as indicated in Figure 34 is not uniform, it has approximately 40% coarse and 60% fines. The coarser fraction of material found at the bench ends and in distant free face. Fines are found close to and in the middle of the bench face. During the project study there was no information provided from any model to predict the fragmentation size and the postulation I made in above text is from visual assessment after the blast. In the proposed blast deign **Kuz - Ram Model** was used as the basis to predict the fragmentation expected after the blast. The new blast design predicted fragmentation results are that the average size of material (X_{avg}) as **23cm** and the characteristic size of material (X_C) as **35.4cm.** It can be summarized that the current blast design produces too much fines that is not good while the new blast design creates uniform material with optimum material size, good for loaders.

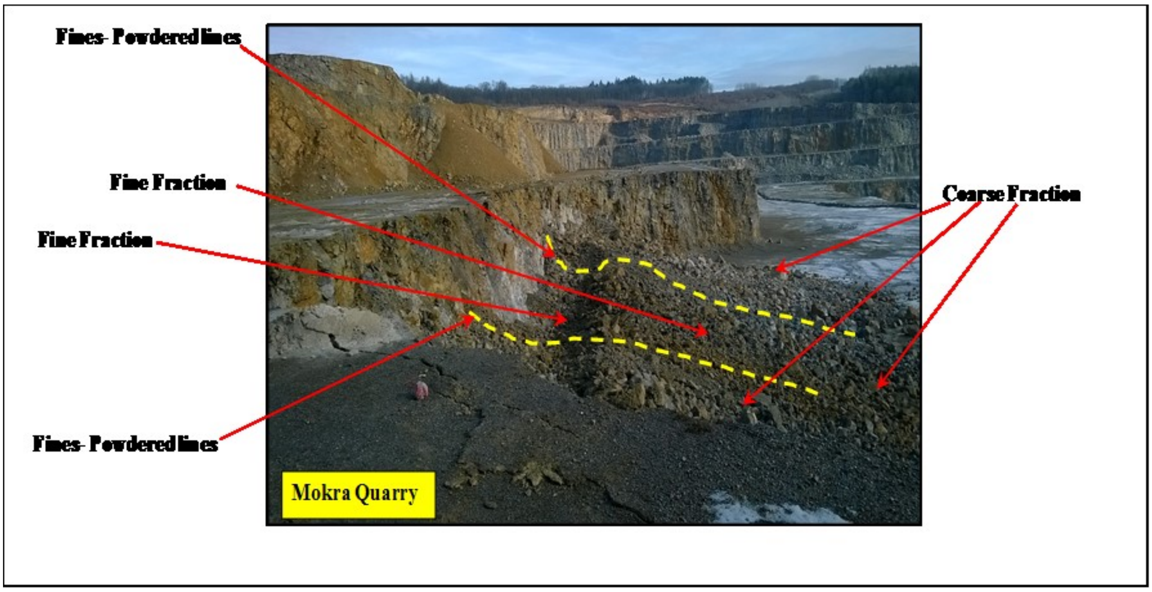


Figure 34: Fragmentation Mokra Quarry (author's field photo)

6.1.4 Secondary Fragmentation

Perusal of fragmentation results of the bench after the blast indicate presence of un-fragmented segments of rock as indicated with numbers **1,3 and 5** in figure 35. This un-fragmented segments of rock will require to be broken and that is called secondary blasting. Secondary blasting could be used to reduce the un-fragmented rock sections into smaller size but it is expensive and time consuming therefor it will slow productivity of the Quarry. The fact that the quarry has to resort to secondary blasting is the sign that their blasting design requires evaluation. Many factors would have contributed to the need of secondary blasting that includes drilling in accuracy, geology, bench geometry parameters, explosive used, blast pattern or the initiation system used. As the proposed blast was not executed the only postulation is that there is no secondary blasting expected therefor reduction in blasting cost.

Figure 35: Un-Fragmented Sections of Rocks (author's field photo)

6.1.5 Economic Benefits

The cost of quarry operation is trade of between drilling, blasting, loading, hauling and crushing. There is clear understanding to minimize the cost and increase the profitability of the operation. In general the distribution of unit cost in a surface is indicated in figure 36. The cost differs at respective mines but as indicated drilling and blasting alone account for 30% of the unit cost, thus it cannot be ignored at all. [16]

The cost of the current blasting operation was said to be **250 000CZK.** It was stated the amount includes labour, drilling, explosives, loading and hauling cost, hence no calculation was provided on how the amount was calculated. Proposed blast design compared to current blasting operation has virtually increase all parameters, thus the anticipation is linear effect between cost and parameters. However the proposed blast design will produce optimum fragmentation thus loading and hauling cost will be reduced. Good blast design will ensure minimum cost for the entire operation and new blast design incorporates that element.

It must be well noted that the objective of the diploma thesis study was not to consider or analyse cost of any form but rather look at which blast design will merely be economically beneficial.

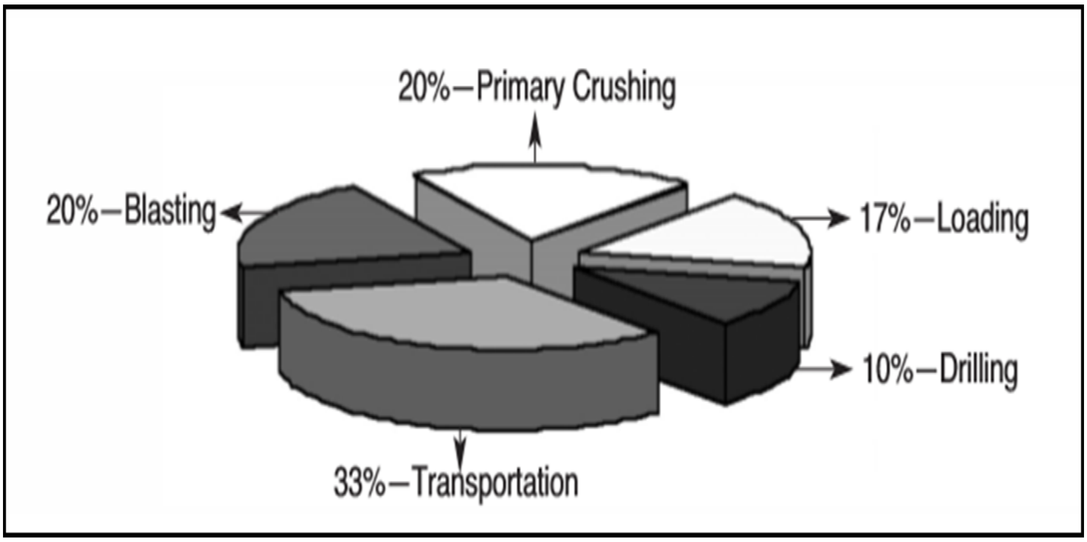


Figure 36: Distribution of Cost in a Surface Mine (%) [16]

7 CONCLUSION AND RECOMMENDATION

Surface Blast design process can be classified as either initial design for planning or optimising design for existing work depending on client, plant or loading and hauling requirements. For many years, the universal approach to designing an effective blast was based on trial and error. [22]

However realization that it is impossible to establish blast design numerically let to establishment of certain empirical rules and equations that form the basis of modern mining blast design practices. [4]

The modernization of blasting works at Mokra Quarry required proper planning and execution to achieve the objectives set as the project commence. During course of the study current blast design and parameters were evaluated, new design was established and comparisons were made to detect defects and initiate remediation measures.

The objectives set before the project commence were **achieved** and detail outcomes will be discussed in this conclusion with recommendation if need possible.

Perusal of fragmentation results from the bench after blasting indicate two main defects, firstly the presence of un-fragmented segments of rock that contributed to uneven face boundary that resulted due to possible high wall damage. Secondly the presence of too many fine material presents a non-uniform size distribution. In general a poor blast is caused by various factors therefor to improve the blast several of those factors will require **Re-evaluation and Adjustment** to obtain a good blast. The existence of un-fragmented segments gives rise to the concept of secondary blasting, that is costly and affect the mine productivity. Drilling as a first step done before other blasting works requires evaluation because if holes are not drilled to proper depth or desired angle meaning in accurate or poor drilling will lead to poor blasting results and vice versa. The current blast defect of un-fragmented segments of rocks and existence of fines cannot be linked with drilling accuracy but it is worth to re-evaluate drilling practice at the quarry. It is highly recommendable to introduce **Drill Monitoring** to improve drill accuracy to aid with good blast results or amend the current staggered drill pattern to square pattern that easier to drill.

The explosives and detonator types used in the current blast design and delay timing are sufficient enough to deliver a good blast results. The proposed blast design incorporates same explosive and detonator types with delay timings because the need to change them was not relevant because they had minimum effect in the poor blast results. However better understanding of geological setting in the quarry would help to change the borehole configuration in terms of amount of energy required to break different type of rock. Current blast design involves blasting limestone and clay rocks with the same energy or powder factor(PDF),therefor blast results show two defects one the powdered lines of fines due to over break and two the un-fragmented segments of rock due weak or demolish applied energy. To eliminate defect of this nature **Decking** is key and a necessity at the quarry that limestone and clay be blasted with correct PDF. This allow fair explosive power distribution and can reduce over break or secondary blasting. The proposed blast design did not incorporate deking because no information was made available with regard to the depths of both limestone and clay rocks on the bench. To calculate explosive loading per borehole required depth.

Fines and un-fragmented segments of rock seen in the blast can also be attributed to poor stemming height and material used. Blast energy can be divided into applied and loss energy. Loss energy manifests itself into excess fines (over blasted material) as seen in the current blast results. General rule at many surface mines is that stemming height is 20 or 30 times the borehole diameter. Using the general rule current blast design gives 1.8m as proposed stemming height, comparing this to the actual stemming height that was used in the blast it is found that it is 1.6m more. Using the same general rule on the proposed blast design gives 2.4, this indicates the proposed stemming is +0.7m which can be defined as optimum. The discrepancies are attributed by the fact the proposed design used larger bore hole diameter. The current blast design requires a change in the **Stemming Height** and an increase in **Drill Diameter** to avoid fines and existence of un-fragmented segments of rock.

Deterioration in muckpile shape parameters as seen in the current blast design implies poor high throw and spreading of muck thus will result to higher dozing hours and more excavation cycles time for the loader. The proposed design eliminates or minimize the dozing hours and excavation cycle time because it used closed chevron V1 firing system and the predicted muckpile profile is of material collected centrally and less spread. This allows the loader to focus on primary task of loading material rather spending time dozing or preparing material for loading. It is highly recommendable to change the current diagonal firing system with **V-type of firing system**. The case study in the literature studies of this thesis can be used as bench mark to support the recommendation made above.

Current blast design of S=B, giving spacing to burden ratio of 1. The spacing to burden ratio of 1 is not necessary bad because the current staggered pattern provides coverage of 98.5%, thus theoretically it signifies a good breaking but the results of the blast show otherwise. The proposed blast design with square burden and S>B, giving spacing to burden ratio of 1.2, thus expected cover of 75% that is 23.5% less. The question then arises why the staggered pattern did produce non-uniform material because ideally it should produce good blast. The poor performance of staggered pattern can be attributed to geology, as the existence of fracture has potential to reduce development of radial fractures. Other factors that may have contributed to poor performance of staggered pattern are small diameter holes at relatively high bench, drilling inaccuracy and hole deviation can result in the pattern at the toe being unrelated to the laid out pattern on the bench, therefor square pattern is preferable. It would be beneficial to change the **current Spacing to Burden ratio** to a relationship of (S>B) because practices at various mines have emphasize the need that spacing be kept no less than the burden(S>B) if not it causes premature splitting of holes and early loosening of stemming column resulting in the sudden drop of blast hole pressure thus can affect fragmentation negatively.

Many fines produce by the current blast design are not ideal. Proposed blast design uses Kuz-Ram model to predict expected fragmentation size after the blast. The predicted average size of material (X_{avg}) is **23cm** and characteristic material size is **35.4cm.** Distribution curve of percent passing shows low MFS K50 value that means uniform and material easy to load, better fill factor for loader and less boulders. The quarry needs to get familiar with the use of **Kuz-Ram Model** or other model as basis to know expected material size prior to blasting. This will help the quarry to optimize the current blast design in terms fragmentation size.

The loading equipment for the current blast design is best suited for Wheel loader due to muckpile profile nature. Wheel loader (FEL), most productive with muckpile profile of that nature. The proposed blast design muckpile profile is best suited for **Hydraulic Shovel** or **Larger wheel loader**. Therefor the proposed design provides an option of two loader possibilities.

The economic aspect of blast design is important and must not only be assessed in isolation but by incorporating other cost of the production cycle. The cost of the current blasting works was stated to be 250 000CZK which is N$163 398.69 with exchange rate of 1N$ = 1.53CZK. Since no information was made available to how that total coat was reached it was not of interest to analyse. The current blast was not big but expectation is it would cost more than the given value because of secondary blasting and underutilization of loader the cost expected must be higher than the amount given. The proposed blast design is characterised by more holes that would require more bulk explosives and more detonators etc. Thus the expectation is a linear effect on drilling and blasting cost to increase however it provides economic benefits.

Economic benefits from proposed blast design can be best summarized as;

- Safe blasting practice with no fly rock issues.
- Provides maximum production blast of bench therefor high tonnage gains thus increase profit/income.
- No secondary blasting – cost savings.
- Minimum high wall damage – good for next blast and improve safety.
- Optimum fragmentation size means;
  - Material can easily be loaded and hauled.
  - Less wear and tear on the equipment, so there will be lower maintenance cost.
  - Better fill factor more tonnage move to the plant.
  - Shorter excavation and haul cycles.
  - Cost of comminution reduced.
  - High tonnage thus increase in production.
  - Reclamation cost reduce because dealing with better fragmented material will make grading less costly. There will be little to no oversized material to deal with and the dozers should be able to manage with minimum cost.

In my closing remarks:

The diploma thesis assignment was interesting for me as a student from Namibia (Africa).It was clear that the comparison of the blast design would not be easy because some data was to my own discretion. However I did use values used in the current industry to get the better judgment as to the blasting works and my visual assessment during the field visit to the quarry guided me during investigation. The mine has done good job in terms of reclamation and that is factor that I would take back home to try change the attitude of mining companies operating in my country. I was surprised that the quarry had no female employees in areas of production but may be it is because I am custom to see a female handling a hydraulic loader or driving an articulated dump truck at mining operation in my country.

I am very thankful to all parties (Technical University of Ostrava, Austin Powder, and Mokra Quarry) that made my visit to the quarry possible for diploma thesis assignment.

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LIST OF SYMBOLS AND ABBREVIATIONS

ADT	Articulated Dump Truck
Al	Aluminium
ANFO	Ammonium nitrate- fuel oil
A.SA	lead azide, lead styphnate and aluminium
B	Burden
BCM	Bank Cubic Meter
$CaCo_3$	Calcium Carbonate
C.B	Control Burden
CZK	Czech Koruna
C.S	Control spacing
Cu	Copper
DNT	Dinitrotoluene
FEL	Front End Loader
g/cc	gram per cubic meter
MPa	Mega Pascal
ms	millisecond
NG	Nitroglycerin
N$	Namibian Dollars
PDF	Powder Factor
PETN	Petaerythrite Tetranitrate
RQD	Rock Quality Designation
S	Spacing
t	Tonnes
TNT	Trinitroluene
VOD	Velocity of Detonation

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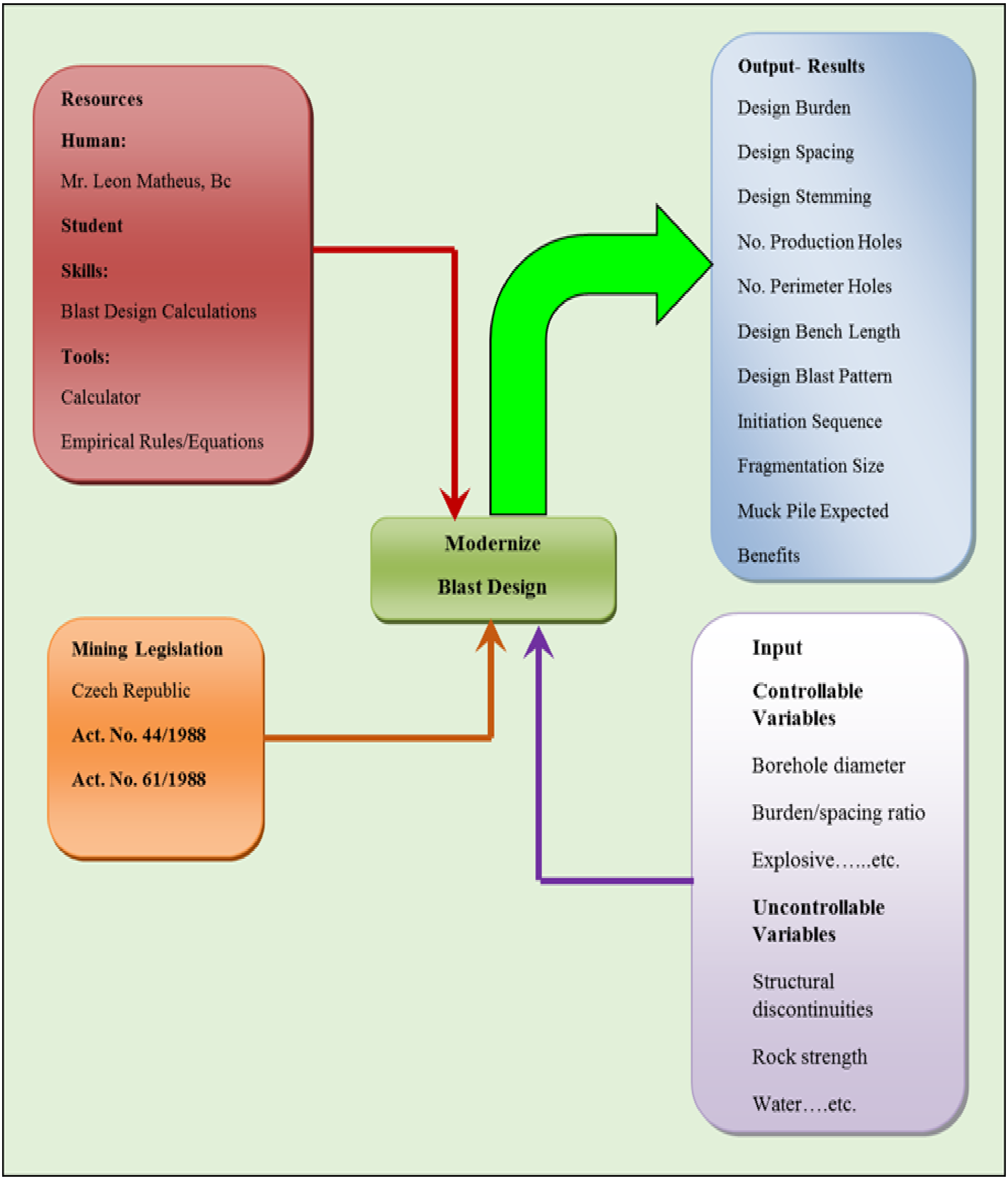
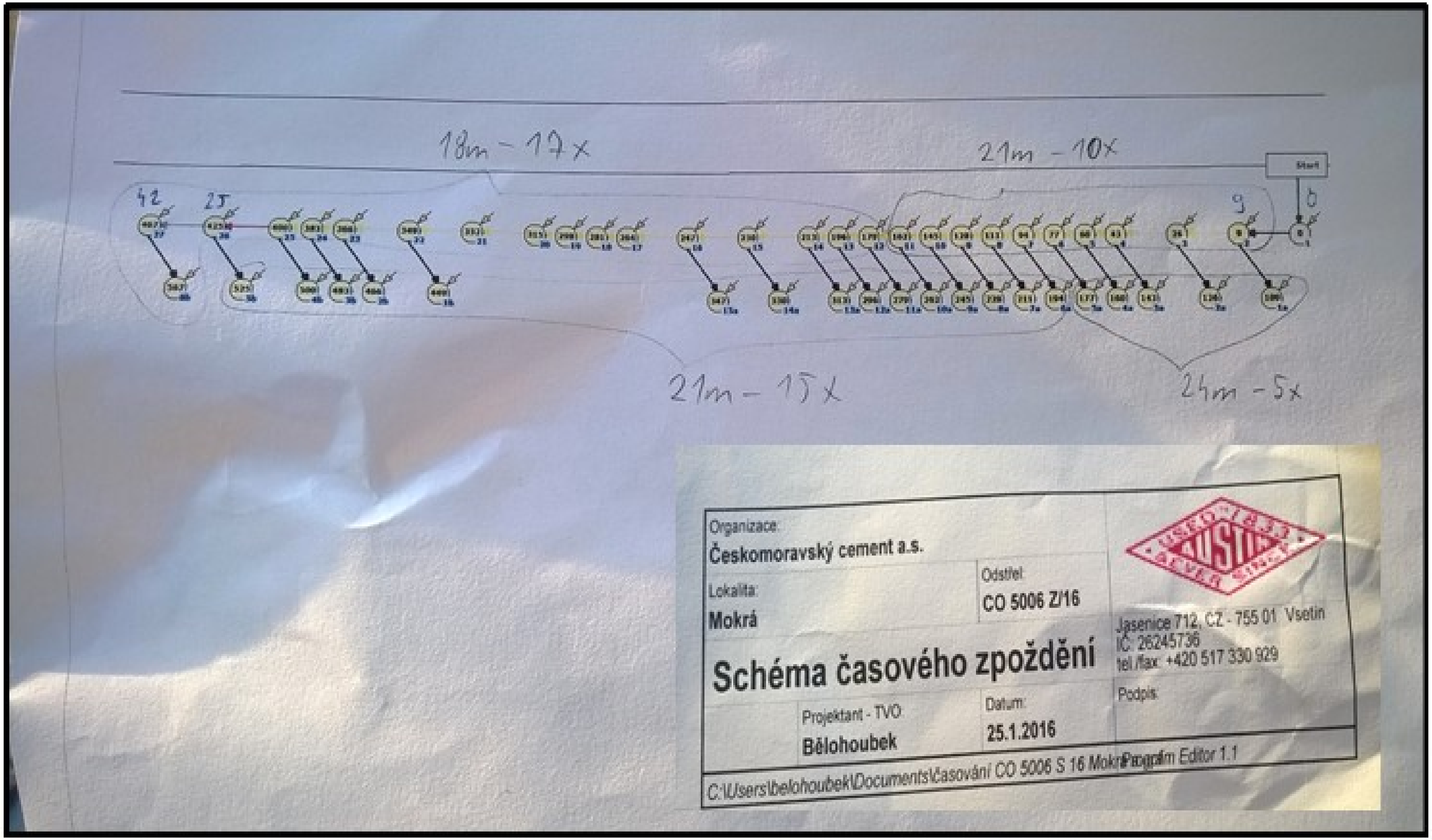
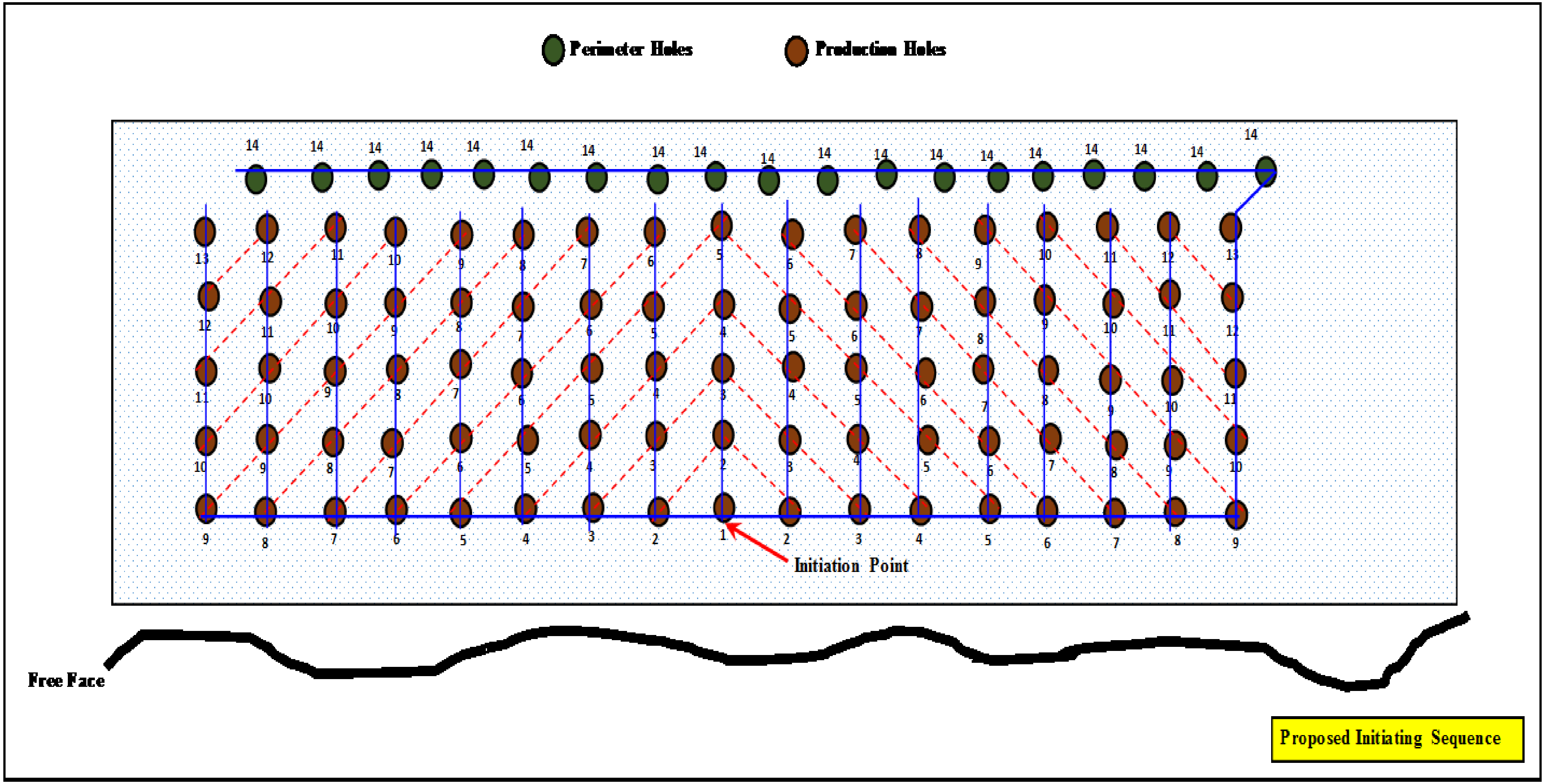
Appendix 1: Project Planning Model (author's illustration)



Appendix 2: Current Blast Design Layout (author's field photo)

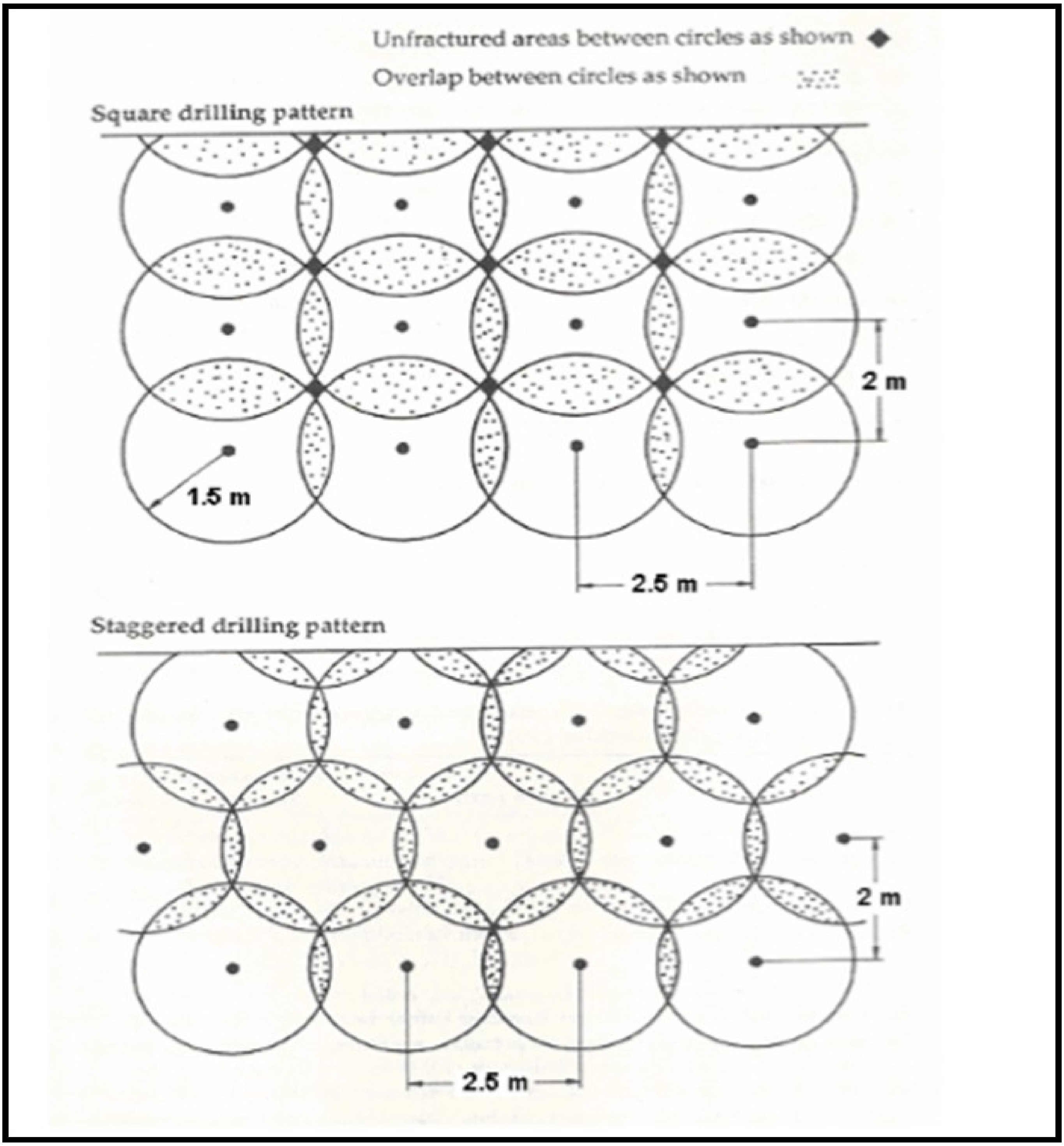


Appendix 3: Proposed Firing Sequence (author's illustration)

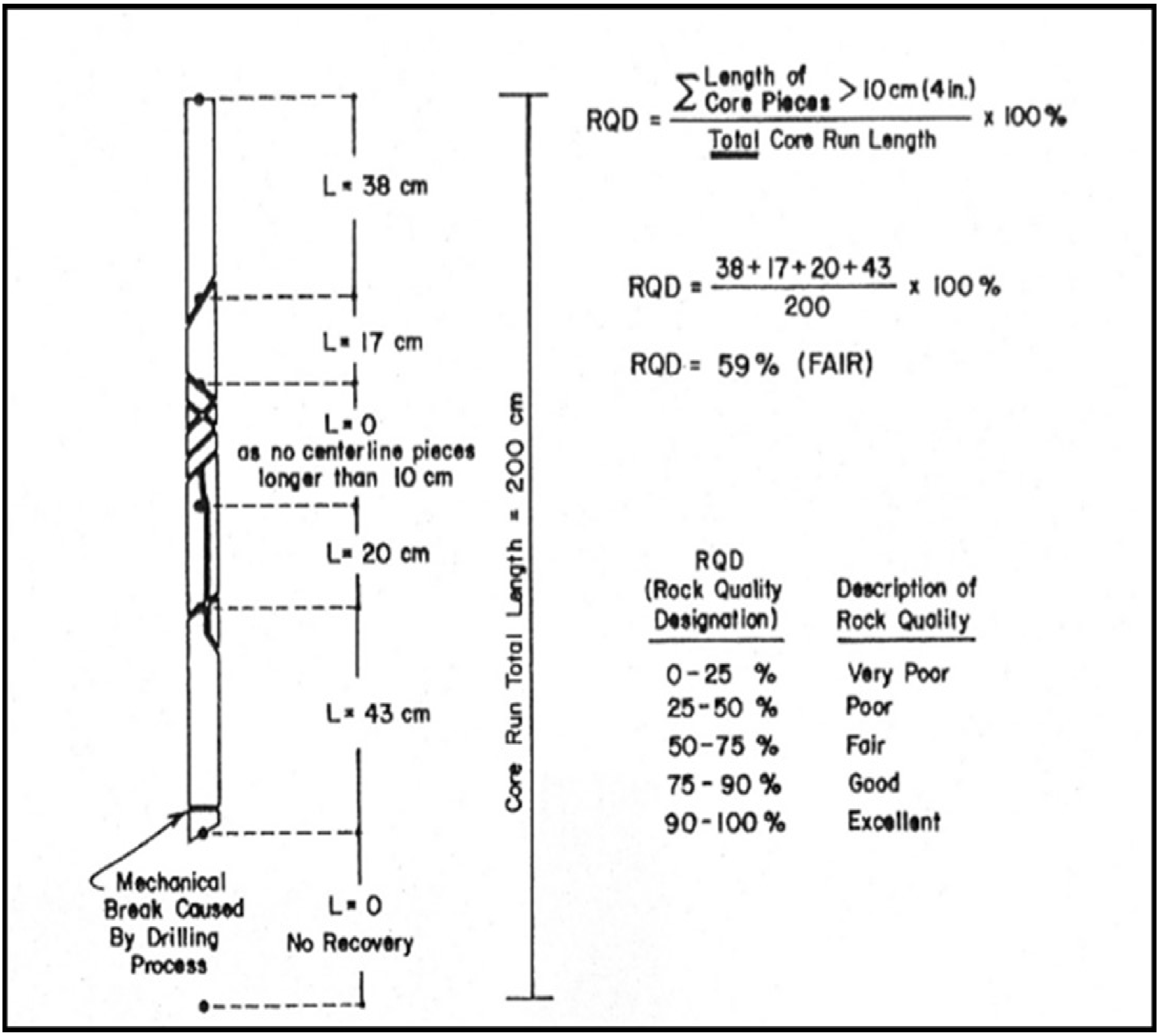


Not drawn to scale;

Appendix 4: Blast Patterns and Coverage [31]



Appendix 5: RQD Selection Criteria [15]



Appendix 6: PDF Selection Table [31]

General Category	Rock type	Powder factor (kg/m^3)	Rock factor A
Hard (+200)	Andesite Dolerite Granite Ironstone Silcrete	0.70	12 -14
Medium (100 – 200)	Dolomite Hornfels Quartzite Serpentinite Schist	0.45	10 -11
Soft (50 – 100)	Sandstone Calcrete Limestone Shale	0.30	8 - 9
Very soft (-50)	Coal	0.15 – 0.25	6

Appendix 7: Western Part of Quarry - Panoramic View of Area Z2 [27]

Appendix 8: Western Part of Quarry - Map Location of Area Z2 [27]